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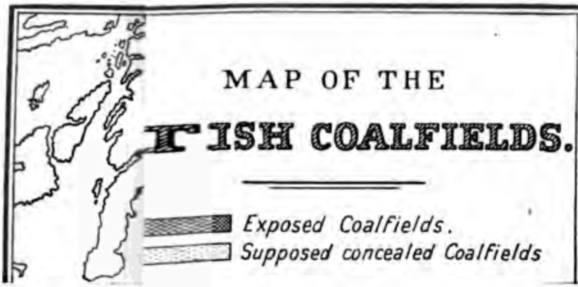
MAP OF THE
ISH COALFIELDS.
Exposed Coalfields.
Supposed concealed Coalfields

AN ELEMENTARY CLASS-BOOK

OF

PRACTICAL COAL-MINING

FOR THE USE OF

STUDENTS ATTENDING CLASSES IN PREPARATION FOR THE BOARD OF EDUCATION AND COUNTY COUNCIL EXAMINATIONS OR QUALIFYING FOR FIRST OR SECOND CLASS COLLIERY MANAGERS' CERTIFICATES

BY

T. H. COCKIN

MEMBER OF THE INSTITUTION OF MINING ENGINEERS
CERTIFICATED COLLIERY MANAGER
LECTURER ON COAL-MINING AT SHEFFIELD UNIVERSITY COLLEGE

With Map of the British Coal-fields and over 200 Illustrations specially Drawn and Engraved for the Work

Capio Lumen

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PREFACE.

The Author's experience as a Mining Lecturer in Yorkshire and Derbyshire leads him to believe (although excellent text-books by experienced writers are already available) that there is still an opening for an Elementary Class-book which shall give a student not only a general grasp of the principles of Coal-Mining, but also some insight into allied subjects such as Chemistry, Mechanics, Steam and the Steam Engine, and Electricity.

The tendency of the times is towards a higher standard in all branches of Technical Education, in consonance with which a higher state of efficiency is now being demanded of those who present themselves for Colliery Managers' and other Mining Examinations. It is with the view of meeting these conditions that the Author has carried this work to a rather more advanced stage than has hitherto been considered necessary for an Elementary Class-book.

In dealing with the various topics of the volume, the Author has endeavoured to start at the very commencement, and has assumed no previous knowledge on the part of the reader; at the same time, obsolete methods have not been described

except where they illustrate principles or point out the trend of modern improvements; and in this way space has been economized, with the result that probably in no text-book on Coal-Mining published at so moderate a price will be found such a complete and advanced treatment of the subject.

While not adhering to the formal syllabus of any mining examination, the Author has covered the ground required by the Board of Education and County Council Examinations, and the student who is qualifying for his First or Second Class Colliery Manager's Certificate will find that this volume will supply him with the theoretical knowledge he needs, and in addition, furnish him with many and varied examples of actual mining practice, drawn from some of the largest collieries, and those best equipped with modern appliances, in the country.

The illustrations in the text are for the most part from original drawings specially prepared for this work, all unnecessary complications being avoided, so that the diagrams may be easily understood and used as examples for sketching.

A large number of arithmetical examples have been introduced, the working out being shown in detail in order to enable students to readily grasp the principles involved.

The Author has not ventured to undertake the preparation of this work without a practical connection with collieries in several counties, extending now over many years; and he cannot too strongly express the opinion that the closest study of even the best text-books, and those of the most elaborate kind, will be of little avail to the aspiring colliery manager unless coupled with practical experience in the mine; and as the conditions of Coal-Mining vary so widely, some

degree of knowledge of as many coal-fields as possible is also desirable.

Finally, the Author has pleasure in acknowledging the assistance afforded him by Mr. HERBERT PERKIN, who has kindly revised the text and made many valuable suggestions. He is also indebted to several manufacturing firms for information given in answer to his inquiries, and, in some instances, for the loan of blocks of illustrations.

SHEFFIELD,
September, 1904.

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ERRATA.

Page 41,	*line* 17,	*for*	"at least" *read* "about."
" 45,	" 7,	"	"Moria" *read* "Moira."
" 76,	" 36,	"	"square foot" *read* "square inch."
" 80,	" 27,	"	"9040" *read* "9048."
" 224,	" 7 & 8,	"	"316,646" *read* "316464."
" 271,	" 18,	"	"16 feet" *read* "21 feet."
" 271,	" 19,	"	"14-feet" *read* "16-feet."
" 274,	" 10,	"	"$3\cdot25 \times 2$" *read* "$3\cdot25 \times 5\cdot2$."
" 284,	" 5,	"	"$F = \frac{C \times 9}{5}$ and + 32" *read* "$F = \frac{C \times 9}{5} + 32$, and."
" 287,	" 21,	"	"32·725 cub. feet" *read* "32725 cub. feet."
" 310,	" 18,	"	"$B = B^2 \times 1\cdot5$" *read* "$B = C^2 \times 1\cdot5$."
" 332,	" 8,	"	"Fig. 156" *read* "Fig. 162."
" 359,	" 2,	"	"$\frac{60 \times 2}{40}$" *read* "$\frac{60 \times 2}{3}$".
" 420,	" 26,	"	"*Polpyhase*" *read* "*Polyphase.*"

intensely hot. That the interior of the earth is extremely hot is proved to some extent by volcanoes, by the presence of igneous rocks, and by the fact that the deeper we penetrate the earth's crust, the higher the temperature becomes.

As the crust of the earth cooled, the atmosphere would be

formed from the gases which surrounded the molten globe; water would also be produced, and the surface of the earth would gradually assume something like its present form.

The gradual cooling of the crust would lead to shrinkage, and consequently pressure, causing the rocks to bend and break. This shrinkage is no doubt still going on, but as the earth gets older, and the solid crust thicker, the interior cools more slowly, and the crust movements become less violent. This, to some extent, explains the fact that the older rocks are always found to be much more crushed and contorted than the newer ones.

It is not possible to make any reliable estimate of the earth's age; but the manner in which the changes in the earth's crust have taken place can only be properly understood when it is remembered that these changes have been spread over an incalculable space of time.

It might be difficult, for example, to understand how rock-beds thousands of feet in thickness have been formed. But the difficulty disappears when it is realized that the addition of only one inch per annum would amount to over fifteen miles in a million years; and the earth is certainly many million years old.

Classification of Rocks. — The rocks (geologically speaking, all beds are termed "rocks," even though composed of clays, sands, coal, etc.) comprising the earth's crust are divided into two great classes: viz. *igneous* and *aqueous*. There is also another division, known as *metamorphic*, or *altered* rocks, which has some of the characteristics of both the stratified and unstratified variety.

Igneous, or Unstratified Rocks.—These are found in large shapeless masses; they are without any regular line of dip or cleavage, and are usually of a crystalline character. The most familiar examples being *granite*, *basalt*, *trap*, and *diorite*.

They appear to have been forced through the earth's crust, to the surface, in a molten condition, and there cooled and hardened. They are termed igneous rocks because they owe their character

to the action of heat. Fig. 1 illustrates the mode of occurrence of the igneous or unstratified rocks. The earth's crust seems to have been rent asunder, and the material which has formed the igneous rock, forced through in a molten condition. In the figure the molten material is shown to have overflown upon the surface, forming a boss and overlying the stratified rocks. At C, the igneous rock is *interstratified*, that is, it has been forced into the stratified rocks, running along the bedding planes; this is frequently the case, especially where the stratified rocks are composed of some soft material such as coal. In some coal-fields much coal is destroyed by the intrusion of igneous rocks in this manner; the molten material having

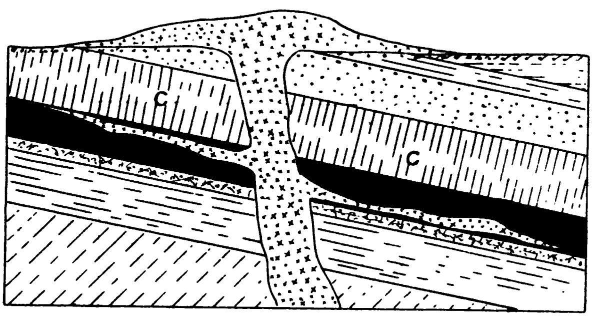

FIG. 1.—Igneous and aqueous rocks.

practically taken the place of the coal over large areas, burning and coking the coal at the points of contact.

Examples of igneous rocks overlying coal-measures, somewhat after the manner shown in Fig. 1, are to be found in Leicestershire and South Staffordshire, in both of which districts shafts have been sunk to coal through a considerable thickness of igneous rocks, which has been forced to the surface and spread over large areas. Unstratified rocks are not very frequently met with in coal-mining; but where they are met with, they are a source of trouble and expense. In some of the largest of the British coal-fields igneous rocks are entirely absent.

Aqueous, or Stratified Rocks.—These occur in parallel layers or strata; they have a definite line of dip and cleavage, and are non-crystalline. Stratified rocks being formed by the agency of water, are known as *aqueous*, or *sedimentary*. All stratified rocks are built up from the waste of older rocks; thus, in order to form new rocks, older ones have to be destroyed. Some of the stratified rocks are built up of the waste of older stratified rocks, others from the waste of igneous rocks. The process by which the stratified rocks have been formed, and by which they are still being formed, is as follows: The existing rocks are continually wasting away, or being *denuded*, chiefly by the action of rain and frost, assisted in some cases by the wind, sea, and running water. The *débris*, or waste material, gradually makes its way down to the streams and rivers, which carry it out to sea, in the shape of mud, sand, and shingle. As the river widens out and enters the sea, its velocity is checked, and the mud, sand, and shingle are deposited at the bottom—the heavier falling first and the lighter being carried further out. In course of time, these beds become solidified by the pressure of the material deposited upon them, and stratified rock beds are formed, the mud, sand, and shingle becoming beds of shale and fine- and coarse-grained sandstone.

Rocks formed in this manner are known as mechanically formed rocks, and embrace the sandstones, shales, clays, etc.; some stratified rocks, however, have been formed *chemically* and some *organically*. Travertine, or tuffa, is an example of a chemically formed rock; springs charged with carbonic acid gas pass through crevices in limestone, and the water, owing to the presence of the carbonic acid gas, carries lime away in solution; but when the spring breaks out into the open air, the carbonic acid gas escapes, and the lime is re-deposited, forming a newer rock bed. Other examples of chemically formed rocks are gypsum, rock-salt, and some limestones.

Organically formed rocks are those built up from the remains of once living vegetable or animal organisms. Some limestone, for example, is composed almost entirely of the remains of marine plants and animals. The plants and animals had the

property of extracting lime from the water in which they lived; as they died, their remains, which consisted chiefly of carbonate of lime, sank to the bottom, and in the course of time built up rock beds of immense thickness. Coal is also an organically formed rock, being composed of the remains of vegetable substances.

Metamorphic Rocks.—Rocks which have been so much changed since their deposition as to have lost all traces of their original character, are known as *metamorphic*, or *altered*[1] rocks. The alteration is, in most cases, due to heat and pressure, which has usually destroyed the lines of stratification and caused semi-crystallization. Some metamorphic rocks merge into igneous, others into stratified rocks.

Age of Rocks.—It is, of course, impossible to arrive at the exact age of any group of rocks; but the relative age of the stratified rocks has been determined. This has been done by ascertaining their relative positions; thus, if one group of rocks is always found above another, it follows that the group which occupies the lower position is the older. Information as to age has also been gathered by studying the fossil remains found in the various rocks, and the character of the rocks themselves.

The relative age of igneous rocks is difficult to determine, but may be roughly estimated by finding the newest rocks which they intersect, and the oldest faults which intersect them. For example, if a bed of basalt intersects coal-measures, it is obviously newer than the coal-measures; and if this basalt was found to be affected by a fault that also affected the coal-measures, the basalt would have been formed earlier than the fault, and later than the coal-measures.

Order of Succession of Stratified Rocks.—The stratified rocks have been divided into four main groups, the classification depending upon the form of life which existed during the period when the groups were being formed, a record of which is preserved in each by the fossils.

These main groups are—

Cainozoic, or Tertiaries	(recent life).
Mesozoic	(middle life).
Palæozoic	(ancient life).
Azoic	(without life).

These main groups are subdivided into the following systems:—

Cainozoic	Post-tertiary. Tertiary
Mesozoic	Cretaceous. Oolitic. Lias. Trias.
Palæozoic	Permian. Carboniferous. Devonian, or Old Red Sandstone. Silurian. Cambrian. Laurentian.
Azoic	Rocks older than Laurentian, mostly Crystalline and Metamorphic.

Each of these systems may be subdivided into rock groups, and the rock groups into beds.

The rocks shown in the table would have a thickness of about twenty miles if they were all present together, but this is never the case. The surface is sometimes composed of the rocks of one system, and sometimes of another. If, for example, the surface is composed of Carboniferous rocks, all the rocks above the Carboniferous system would be absent; the lower systems might, or might not be present, but where they did occur, they would be in their proper order. In short, though rock systems are frequently missing, they are never out of their proper order unless under very extraordinary conditions which are never met with in coal-mining.

The order of succession of the stratified rocks is of the

greatest importance; it shows us, for example, that it would be useless to bore for coal in any district whose surface was composed of rocks belonging to any period earlier than the Carboniferous; whereas, if the surface consisted of rocks higher up in the scale, the Carboniferous system might exist, though it by no means follows that it would, as frequently many of the systems are entirely absent, and rocks high up in the scale are found reposing on rocks quite low down. A detailed description of the whole of the different systems is beyond the scope of this volume, but the economic value of each is briefly as follows:—

Tertiaries.—Clays, building-stones, marbles, sand, gravel, lignites, cement.
Cretaceous.—Chalk, flints, iron ore, inferior coal.
Oolites.—Clay, iron ore (Northamptonshire), building-stone (Bath).
Lias.—Ironstone (Cleveland), limestone, alum shales, jet, inferior coal.
Trias.—Gypsum, rock-salt, clay (Peterborough).
Permian.—Building-stone, magnesian limestone.
Carboniferous.—Coal, ironstone, fireclay, building-stone, limestone, lead, zinc, copper, barytes, chert, umber, manganese.
Devonian.—Marble, iron ore.
Silurian.—Limestone.
Cambrian and Pre-Cambrian.—Gold (Dolgelly), copper, granites, marbles, slates.

The Carboniferous and Adjacent Systems.—The Carboniferous system is overlaid by the Permian, which, in its turn, is covered by the Trias, when all three systems are present. As most of the undeveloped coal in Great Britain lies in measures which are covered by one, if not by both, of these formations, it follows that most new shafts will have to be sunk through them. Both these systems are, therefore, of great interest to the coal-miner.

The Trias.—This system occurs above the Permian, being, in some places, conformable, and, in other places, unconformable with it. The system in Great Britain is divided

into two main series of rocks, viz. the Keuper and the Bunter. The *Keuper* consists of red and green marls and shales, with occasional beds of sandstone. In Cheshire valuable beds of rock-salt are found in this series, and in Nottinghamshire important deposits of gypsum have been worked in it for many years.

The *Bunter* series of rocks consists of soft variegated sandstones and marls with thick pebble beds. Its chief industrial importance is its value as a water-bearing rock, large supplies of excellent water being obtained from bore-holes in it. Its water-bearing qualities make it very costly to sink through.

Rocks belonging to the Triassic system cover a large area in the centre of England, usually forming rich undulating pasture land, with few hills of any importance.

The Permian System.—The rocks forming this system consist of sandstones, marls, and magnesian limestones. In Yorkshire and Derbyshire the Permian rocks form a long narrow band lying along the whole of the eastern boundary of the coal-field, and carrying two beds of magnesian limestone, both of which are extensively quarried for lime-making and building purposes. The stone used for building the Houses of Parliament was obtained from quarries working these rocks.

The Carboniferous System.—The main divisions of the Carboniferous system, as it occurs in Great Britain, are as follows :—

Upper coal-measure.
Middle do.
Lower do.
Millstone grit.
Yoredale shales and limestone.
Mountain, or Carboniferous limestone.

The upper coal-measures consist of shales and sandstones, with thin beds of limestone and thin seams of coal. This

series is not represented in the Yorkshire and Derbyshire coal-field, and is nowhere of great industrial importance.

The middle or true coal-measures are the series of rocks from which the great bulk of our coal is derived. The measures consist of sandstones, shales, and clays, with numerous beds of coal and ironstone. The sandstones vary in texture, but are usually fine grained; their colour is white, yellow, or pale blue, though sometimes stained a reddish tint owing to the presence of iron.

The shales are mostly blue or black in colour, some merge gradually into sandstones, and others are highly bituminous, and may contain a considerable quantity of oil. The clays are usually of a hard character; some contain much silica, and are valuable as fire-clays; other clays are carbonaceous, and contain stigmaria rootlets, this being frequently the case with the "spavin," which is usually found underlying the seams of coal.

Formation of Coal.—There are two theories as to the formation of coal-seams; both agree that coal is formed from the vegetable remains of dense forests, or, more probably, of thick low-lying swamps covered with great masses of luxuriant vegetation.

The difference of the two theories is that, while the one considers that the coal-seams occupy the exact site of the swamps, the other maintains that they do not. The former is known as the *in situ* theory, and the latter as the *drift* theory.

The *in situ* theory appears to be the more generally accepted of the two, but neither accounts in a very satisfactory manner for the whole of the phenomena which present themselves in connection with the formation of coal-seams.

The "in situ" theory.—According to this theory huge swamps or marshes covered the area now occupied by coal-seams, this area subsided, or sank below water-level, owing to the gradual movements of the earth's crust, and was covered with water, which formed the medium for the conveyance of mud, sand, etc. This mud and sand covered the vegetable deposit, and,

by becoming hard and solid, formed beds of shale, sandstone, etc., which, in course of time, filled up the shallow waters in which they were deposited, other marshes of dense vegetation grew on the site, subsidence again took place, more mud and sand was deposited, forming more beds of rock and shale, and so on throughout the whole series.

The "drift" theory.—This theory holds that coal-seams were formed in exactly the same manner as the shales and the sandstones which surround them. According to it, the swamps and forests grew elsewhere, probably on the banks of large rivers, and vast masses of decomposed vegetation were transported by water and deposited in their present position, which may have been a large inland lake, or, more probably, the mouth of some great river.

In considering these two theories, the following questions naturally suggest themselves :—

1. Is coal being formed at the present day? and if so, how? (Will not the large peat-bogs, in course of time, become beds of coal?)
2. Are not the rootlets, which are found in the under-clay, the roots of the vegetation which formed the coal?
3. How is it that coal-seams are so regular in extent and thickness?
4. How are the "bats" and carbonaceous shales formed, some of these being half coal and half dirt, and others being shale at one place and coal at another?
5. All coal-seams are "laminated," that is, they are divided by partings into various bands and qualities. How is this to be accounted for?

It may be that some seams were formed *in situ*, and others by *drift*. It is a subject which presents many difficulties, but the student may form his own opinion by studying the questions suggested with reference to the coal-field with which he is most familiar.

The *lower coal-measures or ganister series* consists of flagstones, shales, and gritstones, with thin seams of coal, ironstone, ganister, and fire-clay.

The ganister, from which the series sometimes takes its name, is a very hard, fine-grained, and highly siliceous sandstone; it contains, in some cases, up to 98 per cent. of silica. In the Yorkshire coal-field the most important seam of ganister is found underlying the Halifax Hard, or Ganister seam of coal. It varies in thickness from a mere trace up to 5 or 6 feet, and is used in the manufacture of refractory bricks for steel furnaces, etc. The lower coal-measures also contain the ironstone from which the well-known Lowmoor and Farnley iron are made.

The millstone grit is best developed in Yorkshire and Derbyshire. It consists of massive grits, made up chiefly from granite *débris*. These grits vary greatly in texture, some being very fine and others quite coarse. In Derbyshire there are four beds of gritstone separated by shale; most of the well-known "edges" of the Peak district are formed by the escarpment of the third bed of grit. Thin seams of coal have been worked from the shales accompanying these grits, and the finer grits make excellent building and ashlar stones, otherwise the millstone grit is of little industrial importance. The coarser grits are made into millstones for grinding oats, cork, etc., hence the name of the series.

The Yoredale and Carboniferous limestones are very well developed in the northern counties, covering a considerable portion of north Derbyshire, Lancashire, Durham, and Northumberland. They are very well represented in the Peak district of Derbyshire, where they form some of the well-known "Dales" around Matlock, Bakewell, and Buxton.

The mountain limestone in Derbyshire consists of massive beds of grey limestone, and attains a great thickness. In the north of England and Scotland the limestone splits up, the bottom group of rocks being known as the Calciferous sandstones, and containing the valuable seams of coal and oil-shale which are extensively worked in the neighbourhood of Edinburgh.

Both mountain limestone and millstone grit are absent in

Warwickshire, Leicestershire, South Staffordshire, and Shropshire, and the coal-measures are found reposing on the very old Silurian and Cambrian rocks.

Fossils of the Carboniferous period.—The coal-measure fossils are very numerous, and consist chiefly of plant remains, some of which attain a great size. Freshwater shells are also

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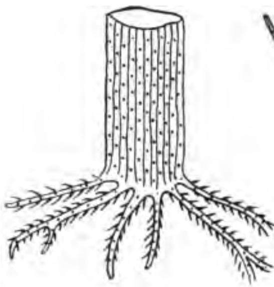

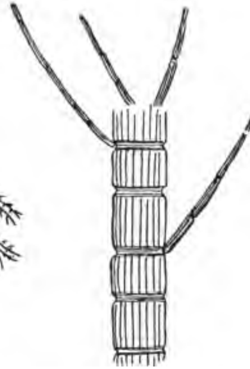

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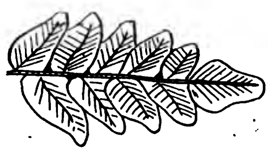

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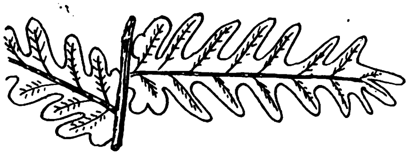


FIGS. 2–5.—2, Sigillaria with stigmaria roots; 3, Neuropteris; 4, Sphenopteris; 5, Calmites.

frequently met with in some districts. The mountain limestone contains a large number of fossils of marine shellfish and plants, whilst the millstone grit contains few fossil remains.

Figs. 2 to 5 show some of the more common fossils which occur in the coal-measures.

Economic value.—The Carboniferous system is of the greatest possible industrial importance, and most of the large centres of industry are found clustered around our coal-fields. From the coal-measures we get coal, ironstone, ganister, fire- and pot-clays, building-stones, and alum shales, all of which are found in beds or seams. From the mountain limestone we get veins of lead, zinc, hæmatite iron ore, barytes, umbers, copper, and manganese, as well as beds of limestone, marble, and chert.

The following is a list of the minerals now being worked in the United Kingdom, abstracted from the Government returns of 1902:—

Alum Shale.—Worked from the middle coal-measures in West Yorkshire from a bed lying between the two seams of the Stanley Main coal. Formerly largely worked from the Lias, near Whitby.

Arsenical Pyrites.—Obtained in small quantities from some of the mines in Cornwall and Devonshire.

Barytes.—From the Silurian and Carboniferous limestones in Northumberland, Shropshire, Durham, etc. Used as an adulterant of white lead.

Bauxite.—Mined from between sheets of Tertiary basalt in county Antrim, Ireland. Used for the manufacture of aluminium.

Bog Ore.—From open works in Ireland. An ore of iron used in the purification of gas.

Chalk.—Used for the manufacture of Portland cement; very largely worked in Kent and Essex.

Chert.—Used in the manufacture of porcelain, and mined from the Carboniferous limestones of Derbyshire and Flintshire.

Clay.—Brick-clays occur in most districts, and fire-clays are chiefly wrought from the middle or lower coal-measures.

China Clay.—Derived from disintegrated granite, and worked in Cornwall and Devonshire.

Coal.—See Chapter III.

Copper Ore.—Worked from veins in Cornwall, Devonshire, and Wales.

Fluor Spar.—Worked in Derbyshire and Durham. Used in making ornaments.

Fuller's Earth.—Worked in Bedfordshire.

Ganister.—Worked in South Yorkshire from the lower coal-measures. Used in the manufacture of refractory bricks.

Gold Ore.—Worked from the lower Cambrian rocks in Merionethshire. In 1902 the value of gold worked amounted to £12,621.

Gypsum.—Occurs in the Keuper division of the Trias, and is mined in Cumberland, Nottinghamshire, Staffordshire, etc. Used in the manufacture of plaster of Paris.

Igneous Rocks.—Quarried in Leicestershire, Aberdeen, Ireland, Wales, etc. Employed for paving, building, and monumental purposes.

Iron Ore.—Quarried in Lincolnshire, Northamptonshire, and Leicestershire from the inferior oolite, and mined in Cumberland and Lancashire (hæmatite) from the Carboniferous limestones; also worked from the middle coal-measures in Scotland and Staffordshire.

Iron Pyrites.—Small quantities picked out at some coal-mines; worked also in Ireland. Used in the manufacture of sulphur.

Lead Ore.—Worked in many places, but chiefly from the Carboniferous limestone in the Isle of Man, Derbyshire, Durham, and Flintshire.

Limestone.—Quarried in most counties, and mined in Scotland, Wilts, Staffordshire, etc.

Manganese Ore.—Small quantities worked in Devonshire and Wales.

Mica.—Obtained as a by-product in the preparation of China clay in Cornwall.

Ochre, Umber, etc.—Obtained from many localities. Used in the manufacture of paint.

Oil Shale.—Largely worked in Edinburghshire and Linlithgowshire from seams in the Calciferous sandstone at the base of the Caboniferous limestone.

Petroleum.—Small quantities are occasionally found in coal-mines.

Phosphate of Lime.—Largely wrought at one time, but now only worked on a very small scale.

Salt.—Chiefly produced from brine in Cheshire, Durham, Lancashire, etc.

Sandstone.—Found in most localities, and chiefly used as building-stones.

Slate.—Largely mined and quarried in Wales, Argyll, Cumberland, and Westmoreland.

Sulphate of Strontia.—Dug from shallow pits in the Keuper beds of Gloucestershire and Somersetshire.

Tin.—Mined from veins in the granites of Cornwall.

Uranium Ore.—Worked at one mine in Cornwall.

Wolfram.—Found in the tin mines of Cornwall.

Zinc.—Often accompanies lead ore. Worked from the Palæzoic rocks in Cumberland, Wales, Isle of Man, Derbyshire, etc.

CHAPTER II.

STRUCTURE OF STRATIFIED ROCKS.

Stratified Rocks.—The stratified rocks were originally deposited horizontally, but they are now usually found lying at a more or less steep inclination. This is due to the pressure to which they have been subjected owing to the shrinkage and movements of the earth's crust. The older rocks are usually found to be much more disturbed than the newer ones, as they have been longer exposed to pressure, and the disturbing influences were probably more violent in the earlier history of our globe.

The main result of the pressure has been to force the rocks into a series of folds. Fig. 6 shows strata folded by pressure.

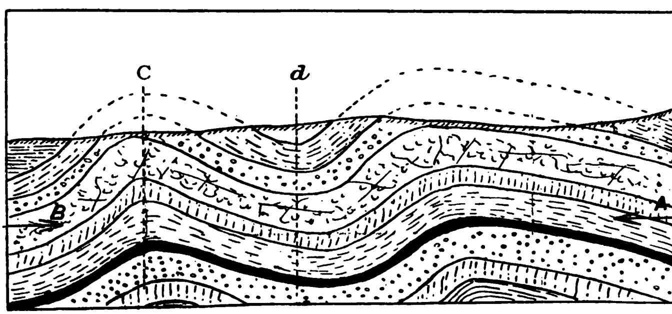


FIG. 6.—Production of folds.

At one time the strata were flat, but enormous pressure at A and B in the direction of the arrows has gradually forced A and B towards each other, bending the rocks as shown, the

dotted lines above the surface indicate the position of the beds after being curved, but the dotted portion has been worn away or denuded. At the point C (Fig. 6) the beds dip away on both sides, forming what is known as an *anticline;* and at *d* the beds on either side dip towards each other, forming a *syncline*, the centre in either case is known as the *axis.*

The size of these folds varies very greatly, they may be only a few feet across or they may be many miles.

Pressure exerted from the east and from the west would result in the formation of a series of folds having their axes running north and south, whereas pressure exerted from the north and south would produce folds lying the other way. Hence, if pressure were exerted from all sides, a double series of folds would be formed, running at right angles to each other; the result of which would be the production of *basins*, in which the beds would outcrop all round and dip towards a central point. Most of our coal-fields are found in basin-like deposits, though in many cases parts of the basins are hidden by the rocks of a later period.

Strike, Dip, and Outcrop.—The inclination at which beds lie is extremely variable; in some cases beds are found vertical or even inverted, and in other cases horizontal. These variations may take place in short distances, but, on the other hand, the dip is frequently found to be very regular over large areas. As a general rule, beds that lie at a high inclination near their outcrop become flatter as their depth increases.

The Strike of a seam is its level line, and the full dip must be measured at right angles to the strike. The inclination of a seam may either be expressed in degrees (thus a seam is said to dip 10° when it makes an angle of 10° with the horizon), or it may be expressed as a vertical fall in a horizontal distance, *e.g.* if the seam dips 6 inches vertically for every yard measured horizontally, its dip would be said to be 6 inches to the yard, or 1 in 6.

Dip may be measured with a clinometer, one type of which is shown in Fig. 7. It consists of two brass straight-edges

hinged together, on the upper is fixed a spirit-level, and on the lower a quadrant divided into degrees, from which the angle made by the two pieces of metal can be read off. To measure an incline with this instrument, the lower limb is placed upon the seam, or upon a long wooden straight-edge, to obtain the average dip, and the upper limb is opened on the hinge until the bubble of the spirit-level is in the centre of its run; the angle is then read off from the quadrant.

Another method of measuring inclinations is that shown in Fig. 8, that is, by means of a straight-edge 10 feet long and an ordinary spirit-level. One end of the straight-edge is raised until the spirit-level indicates that it is level, then the height that it has been raised is measured, and the measurement gives the amount of fall in 10 feet. For example, if the fall is

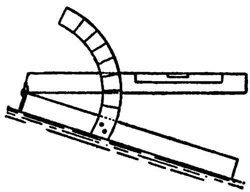


FIG. 7.—Clinometer.

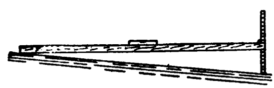


FIG. 8.—Measuring inclination with straight-edge and level.

10 inches, the rate of inclination is 10 inches in 10 feet, that is, 1 in 12.

It is often necessary to convert dip measured in degrees into a vertical rise or fall in a horizontal distance. This may be done approximately by dividing the number of degrees into 57·3, which gives the horizontal distance for a fall of 1. Example—

$$5^\circ = 1 \text{ in } \frac{57{\cdot}3}{5} = 1 \text{ in } 11{\cdot}4 \text{ nearly.}$$

This rule becomes incorrect when the inclination is more than about 20°, and a more accurate method is to take the cotangent of the angle which gives the horizontal distance for a fall of 1. Example: The cotangent of 28° is 1·88, hence 28° dip corresponds to an inclination of 1 in 1·88.

Further information on this subject is given in Chapter XVII.

In measuring the dip of a seam, care must be taken to make the observations along the line of *full* dip, that is, at right angles to the strike, otherwise the dip will appear to be less than it really is. This will be understood by reference to Fig. 9, where *ab* represents the line of strike, and *cd* the full dip. If the dip of the measures is 6 inches per yard, and *cd* measures exactly 1 yard, the end of the line *cd* will be exactly 6 inches below *ab*. Suppose now that the inclination is measured along *cf* instead of *cd*, the point *f* is higher up than *d*, so that the dip per yard in that direction is less than 6 inches. The direction of dip, as well as the rate of inclination, is always required. Thus a seam may dip N. 62½° W., at the rate of 3 inches per yard. On geological and other plans the dip is shown by arrows, thus, ↘ 8°, the arrow head always points downhill, and its direction shows the correct line of full dip.

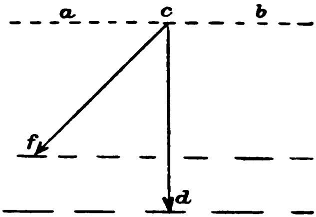


FIG. 9.—True and apparent dip.

The Outcrop of a seam is the edge exposed to the surface. If the surface is level, the outcrop is a straight line, and coincides with the line of strike, otherwise it does not. The form of the outcrop of a bed depends chiefly upon the contour of the surface; hence outcrops may be seen running round hills and along valleys.

The width of an outcrop depends not only upon the thickness and inclination of the bed, but also upon the surface contour. This will be understood by reference to Fig. 23, where the outcrop of the rock bed at *a* is much wider than at *b*, although the thickness and dip of the bed are the same in both cases.

It is difficult for an untrained eye to trace outcrops, but the outcrops of all the principal coal-seams have been mapped by the geological survey for the whole of their exposed length,

and in most cases with great accuracy. Coal-seams are always very inferior at the outcrop, being usually found as a mere bed of smudge.

Bedding and Joints.—Stratified rocks are made up of *beds*, and beds of *layers* or *laminæ*. These laminations cause the rocks to split more or less readily along the bedding planes. The vertical divisional planes, which occur in most stratified rocks, are called *joints*. These are not nearly so regular as the horizontal bedding planes, and are much more strongly marked in some beds than in others. Two sets of vertical joints running at right angles to each other traverse some rocks and cut them up into cubical blocks. Joints or backs are strongly marked in some coal-seams, and very slightly in others.

Conformability.—Beds lying parallel to each other are said to be *conformable*, the beds in the same series are always conformable to each other; but when rocks of different periods come together, the one series may have quite a

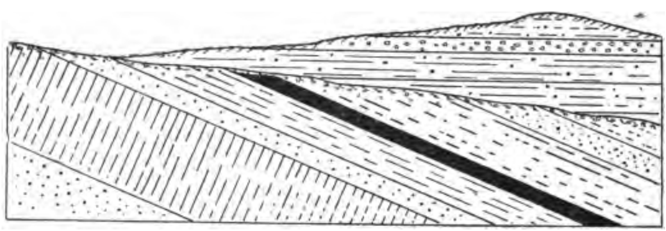


FIG. 10.—Unconformability.

different dip to the other, as shown in Fig. 10, in which case they are said to be *unconformable*. The coal-measures are sometimes overlain by the Permian rocks, the two being unconformable.

Irregularities of Stratified Rocks.—Seams of coal, and all other stratified rocks, are subject to many irregularities

and disturbances. These may be divided into two classes, (A) Those which were caused during the time the seams were in process of formation, and (B) Those which were caused after the seams were formed. It is important to distinguish between these two classes, as those which come under class A usually affect one seam only, whilst those in class B generally affect the whole series of seams.

Irregularities caused contemporaneously with the Formation of the Coal.—*Splitting.*—Coal-seams are much given to splitting and coming together; sometimes the split-off portions of two adjacent seams unite and a third seam is formed, lying between the two original seams and composed of the upper part of one seam and the lower part of the other. One of the best examples of splitting is found in the ten-yard coal of South Staffordshire, which splits up into nine distinct seams of moderate thickness in the northern part of the coal-field. The Silkstone seam in South Yorkshire usually contains a band of dirt about a foot in thickness, but a mile or two north of Sheffield it splits into two seams, with about 30 yards of strata between them; still further north these two seams reunite, forming one seam.

The Barnsley Bed is 10 feet thick in the neighbourhood of Barnsley, but near Sheffield it is only half that thickness; the thinning being chiefly due to the fact that layers of coal, which are a part of the seam in the Barnsley district, have split off and are some distance from the main bed in the roof or floor. Many other examples might be given; in fact, splitting is more or less common in all coal-fields.

Rock faults, or horsebacks.—These usually occur in seams having a rock roof; examples on a large scale may be seen in the Parkgate and Deep Hard seams of Yorkshire and Derbyshire. A cross section of a small example is shown in Fig. 11. From this it will be seen that the roof has gradually come down until it almost touches the floor. Some of these rock faults are small, being only a few feet across and taking only a portion of the seam; but others are very large, and may take

the whole of the seam out for a width of several hundred yards and extend for some miles. The Dumb fault at Alfreton washes out the Deep Hard seam for a width varying between 200 and 500 yards, and has been proved to extend for a length of several miles from the outcrop of the seam.

Rock faults appear to have been caused by a river or stream flowing through the coal-seam just after it had been deposited and whilst it was still in a plastic condition. This has eroded or cut out the coal-seam, and filled in the space with sand and mud, which has in course of time hardened and become rock. In some seams rock faults are quite common, in others they are never met with; they only affect the one seam.

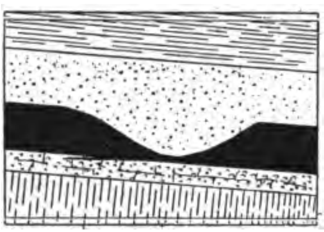


FIG. 11.—Rock fault.

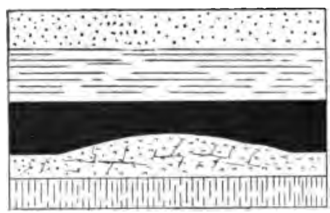


FIG. 12.—Roll.

Rolls.—As shown in Fig. 12, a roll consists of a mass of rock taking the lower part of the seam out. Rolls are probably due to the coal-seams being deposited on an uneven surface or floor. Small rolls are fairly common, but large ones are seldom met with.

Thinning Out.—Some coal-seams and rock beds thin very rapidly and even disappear altogether. This may be due to their being deposited on a sloping floor, or to a failure in the supply of the material from which they were formed. As a rule, coal-seams are more "persistent" than the rocks which accompany them; but the thickness of some coal-seams varies very greatly over quite a small area.

A Swelly or Swilly is an abnormal thickening of the coal-seam. This may be caused by a depression of the surface upon which the coal was deposited, as shown in Fig. 13, in which case the increase in thickness will be from the bottom. Swillies are frequently due to the plastic material forming the coal-seam being partly washed away in one place and piled up in another; or to extraordinary pressure on one part of a seam squeezing the bed thin in one place and forcing the material to another.

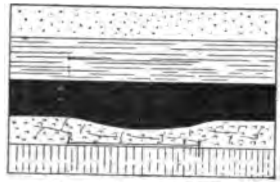


FIG. 13.—Swelly.

Overlap of Strata.—This occurs when a system of rock beds is deposited on a sloping surface. In Fig. 14 the top seam A *overlaps* the lower seam B, owing to the ancient

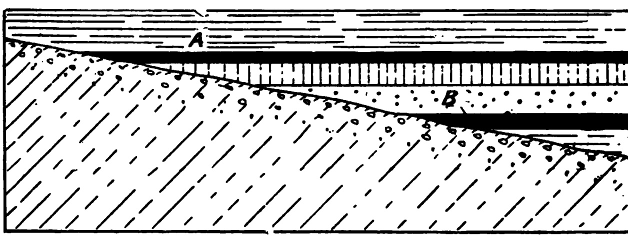


FIG. 14.—Overlap of strata.

hill upon which the coal-measures were deposited. This overlap is of great importance, as it shows that the top seams might be present and the bottom ones absent; it may have considerable bearing upon the extent of some of our hidden coal-fields.

Irregularities caused long after the Formation of the Coal-seams.—*Faults.*—The most important irregularities to which coal-seams and other stratified deposits are liable are

"faults." A fault is a fracture in the measures, usually running at a slight angle from the vertical, and throwing the measures on one side either up or down from their original position. The line of fracture is usually very smooth and polished, and the fissure, when one exists, is usually filled in with *débris*, or "fault muck" as it is termed. Most faults have not been caused by a violent upheaval, but by the gradual movement of the earth's crust; the displacement they cause varies from an inch or two up to many hundreds of feet. Mineral veins are generally the contents of faults. This, however, is very rarely the case in the coal-measures or newer systems, but frequently in the Mountain, Limestone, Cambrian, and older systems.

Faults may continue in almost a straight line for many miles, with little variation in the amount of "throw," but more often the displacement varies, a large fault gradually diminishing and dying away. Sometimes a fault which is an upthrow in one place will change to a downthrow in another. Large faults frequently have smaller ones branching off them in all directions, and often bend about in an apparently erratic manner.

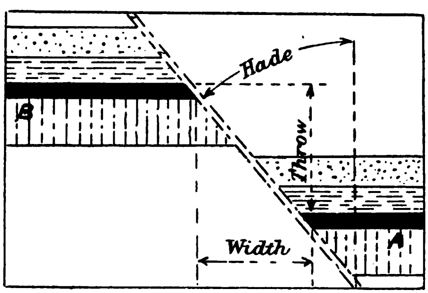


FIG. 15.—Normal fault.

The larger faults frequently run roughly parallel to the line of hills and valleys in the district.

Fig. 15 shows an ordinary or *normal* fault of small displacement. The *hade* is the inclination the fault makes with the vertical, and is expressed in degrees. The average hade of a coal-measure fault is about 20°.

The *width* is the horizontal distance between the fractured ends; it depends upon the size of the fault and upon its hade. The larger the throw and the greater the hade, the greater will be the width.

The *throw* or size of a fault is the vertical distance between the fractured ends, that is, the amount by which the seam has been displaced from its original position. If the fault shown in the sketch were approached from B, it would be called a downthrow, whereas, if struck first from A, it would be said to be an upthrow.

It is very important to notice that if the seam is thrown down, the fault is first hit in the *roof* of the working; but if the fault is an upthrow, it is first struck in the *floor*. So that by observing the direction of the hade of a normal fault it is easy to see whether the coal should be sought above or below. When proving coal across a fault by boring, it is very necessary to drive in right across the width of the fault before commencing to bore, otherwise the seam could never be hit. Quite erroneous results have sometimes been arrived at by not driving in far enough before boring, as it is impossible to estimate the exact width of a fault unless both size and hade are known.

Step Faults.—Faults do not always affect their full amount of displacement in one operation, but often throw the coal up

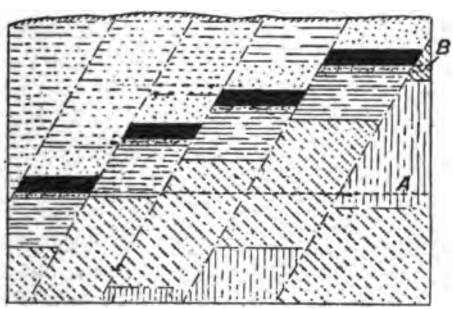


FIG. 16.—Step faults.

or down in a series of steps. In Fig. 16 the total displacement is the vertical distance AB, the coal being thrown down by a series of faults in the same direction, these are termed step faults. A large fault often changes into a series of step faults,

and sometimes even into a *bend* in the strata, which are not fractured, the displacement taking the form of an extremely steep dip in the measures.

Trough Faults.—These are formed by a pair of parallel faults dipping towards each other. They may have been caused by the strata breaking in two places at the top of a fold and allowing a wedge-shaped mass to slip down. The strip of coal is wider in the upper seams than in the lower, and the two faults may run each other out altogether deep down, as shown in Fig. 17; or if one of the faults is larger than the other, as is commonly the case, the displacement in the beds below their intersection will be equal to their difference; thus, if one fault were 70, and the other 60 yards, the displacement below their intersection would probably be 10 yards only.

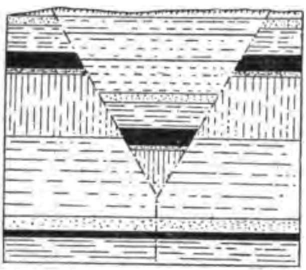


FIG. 17.—Trough fault.

Reversed or Overlap Faults.—This class of fault is very rare in the coal-measures, but may be frequently seen in the rocks belonging to the older systems, where the strata have been subjected to much greater pressure. In reversed faults the strata on one side of the fracture have not only been lifted above the strata on the other side, but have also been pushed over them, thus doubling the beds for some distance, as shown in Fig. 18. The rule as to finding the direction of the throw of a fault from its hade, as given on page 25, has to be reversed when it is applied to these faults, but they are so seldom met with that it is fairly safe to assume that any fault met with in coal-mining is "normal," and not "reversed."

The best-known example of a reversed fault is the "slide fault" at Radstock, in Somersetshire, where the vertical throw is 72 yards, the top seams being doubled for 120 yards, and the lower beds for 330 yards. It should be noticed that the beds on either side of a normal fault are *pulled* asunder, causing

a strip of barren ground, but when the fault is reversed the beds are *pushed* over each other. In other words, normal

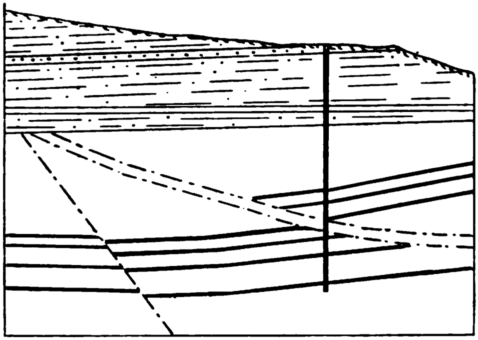


FIG. 18.—Reversed trough.

faults appear to have been caused by a *tensional* strain, and reversed faults by a *compressive* strain.

Effect of Faults on Outcrops.—When a fault runs parallel to the strike of the measures, it is called a *strike* fault. The effect of a downthrow may be to double the outcrop, as shown in Fig. 19. An upthrow strike fault may throw

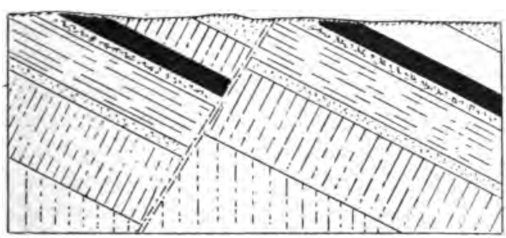


FIG. 19.—Strike fault repeating an outcrop.

a seam of coal out altogether, so that it will not outcrop at all in the neighbourhood of the fault.

A fault running parallel to the dip of the measures is

termed a *dip* fault. When a fault crosses the outcrop of a seam, such outcrop is broken or dislocated, the amount of the dislocation depending upon the inclination of the seam and the size of the fault. It is of great importance to understand this, as the direction and size of the throw of a fault may often be calculated from its effect upon the outcrop of a seam. Fig. 20 shows diagrammatically the plan and section of a fault cutting an outcrop; in the figure the fault throws the measures down to the north, and the outcrops are shifted towards the rise of the beds. Had the fault been a riser or upthrow north, the outcrops would have been shifted towards the dip.

Intersection of Faults.—When two faults intersect one

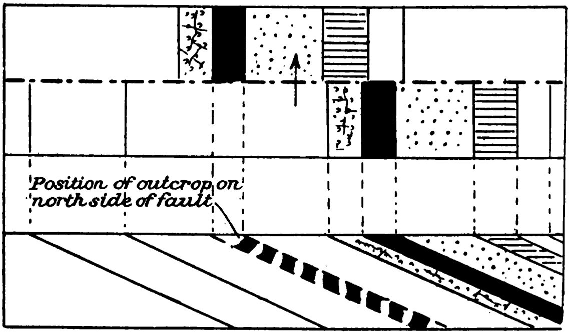


FIG. 20.—Plan and section of fault cutting an outcrop.

another, the line of the older fault is broken at the point of intersection; the same reasoning applying as in the case of a fault intersecting an outcrop.

Dykes.—The strata are sometimes intersected by masses of igneous rock, as shown in Fig. 1, these are termed "dykes." They have the appearance of walls of granite or basalt, and the coal is coked or charred for some distance on either side, proving that the material forming the dyke was forced through the strata whilst in a molten condition. Dykes do not usually displace the coal at either side. Some districts are much affected by dykes, others not at all; one of the best-

known examples is the great whin dyke in the north of England, which can be traced for scores of miles.

Wash-outs.—Fig. 21 shows the Team wash-out. Several seams have been denuded, evidently long after they were formed, a large valley being scooped out and filled in again with boulder clay, probably by glacial action. The depth of boulder clay is about 300 feet, and the width across the top of the old valley varies from about 600 to about 800 feet.

Variations in Inclination.—In most of the larger coal-fields the dip is regular over large areas; but occasionally great variations exist.

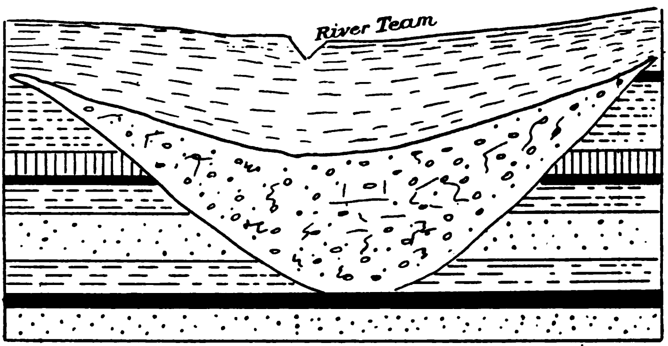


FIG. 21.—Team wash-out.

When the strata are very much twisted, they are said to be *contorted.* Contorted strata are uncommon in the British coal-fields, but are frequently met with in France and Belgium.

Geological Maps.—The geological features of a district are shown upon geological maps. These maps show by colours the period to which the strata forming the surface belong. They also show the position and size of all the known faults, the rate and direction of inclination of the strata, and the outcrop of any beds or veins. In addition to this, they should have marked on them the depths of all shafts and

boreholes, and any other geological information which may be obtainable.

Fig. 22 shows a geological map delineating on a small scale a part of a coal-field. On the west will be seen the millstone grit dipping to the east, and bounded on its eastern side by the coal-measures, the latter in their turn being overlaid by the Permian, and the Permian by the Trias. The information given on this map will perhaps be more clearly

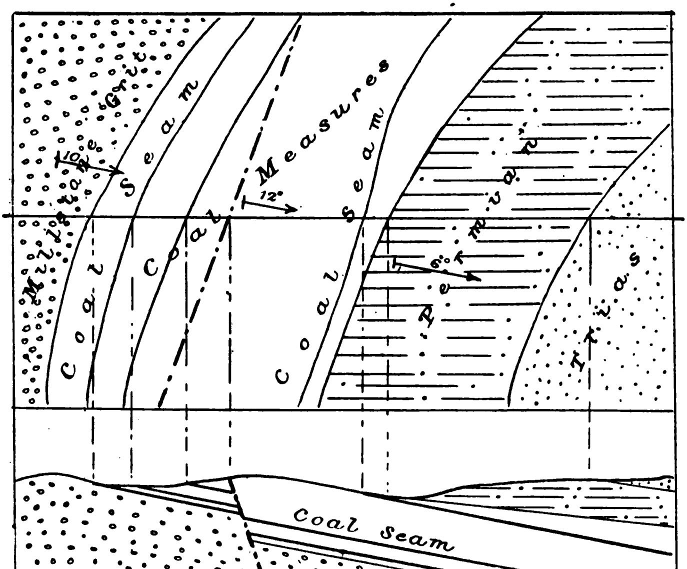


FIG. 22.—Geological plan and section.

understood by reference to the section which has been constructed from the plan. In preparing this section the contour of the surface is first drawn. This is done by making use of the contour lines, which are shown on all the geological maps drawn to a scale of six inches to the mile; these contour lines are level lines marked on the maps, every twenty-five feet, measured vertically, the height above sea-level being marked

on each. The boundaries of the various rock-groups are then projected from the plan as shown by the dotted lines; and the rate of dip of the beds is laid down on the section, as given by the arrows on the plan.

It will be noticed that the millstone grit is shown to extend under the coal-measures, and the coal-measures under the Trias, the whole length of the section. There is no actual proof of this, but it is reasonable to assume that they do so, and in geological sections the lower beds of a series are usually assumed to follow the upper, unless there is evidence to the contrary.

On examining the plan and section, it will be observed

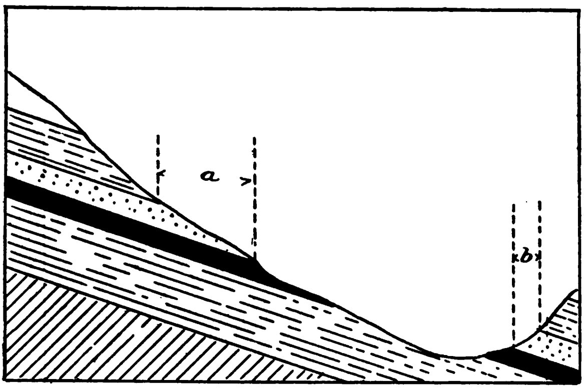


FIG. 23.—Outcrop caused by valley.

that when the surface is level, and the beds not faulted, the boundary of a bed, on its rise side, is its outcrop, and its dip boundary is formed by the edge of the beds which overlie it. It should be noticed that if the surface is hilly, the outcrop may be to the dip of a bed, this will be understood by reference to Fig. 23, which represents an outcrop caused by the erosion of the strata by a valley.

From this it will be understood that the geological information given on plans must be studied in conjunction with the surface contour lines, otherwise erroneous conclusions may be arrived at.

The Government have published geological maps of the whole of England and Wales, on a scale of one inch to a mile, and of some districts, on a scale of six inches to a mile. These maps and the sections which accompany them will be found of the greatest value in studying the geology of any district.

CHAPTER III.

COAL AND COAL-FIELDS.

THE following table shows the great increase in the output of coal from Great Britain and Ireland which has taken place during the last fifty years :—

1854	...	...	...	64,661,401	tons.
1860	...	...	...	80,042,698	„
1870	...	...	...	110,431,192	„
1880	...	...	...	146,969,409	„
1890	...	...	...	181,614,288	„
1900	...	...	...	225,181,300	„
1901	...	...	...	219,046,945	„
1902	...	...	...	227,095,042	„
1903	...	...	...	230,334,469	„

From this table it will be seen that the output is now more than $3\frac{1}{2}$ times as much as it was fifty years ago. Almost every year shows an increase on the preceding one; the year 1901 shows a decrease, which is accounted for by the fact that that year was a period of exceptional prosperity and of high wages, the latter being usually accompanied by a decreased output per man employed.

Produce of Coal-seams.—The specific gravity of ordinary bituminous coal averages about 1·28. The specific gravity of a substance is its weight as compared with an equal bulk of water; and as a cubic foot of water weighs 62·5 lbs., the weight of a cubic foot of any substance is obtained by multiplying its specific gravity by 62·5. Thus the average weight of a cubic foot of coal is $1{\cdot}28 \times 62{\cdot}5 = 80$ lbs.

The weight in tons of an acre of coal 1 foot thick is $\frac{4840 \times 9 \times 80}{2240} = 1555{\cdot}7$ tons.

This quantity is, however, never realized, as there is always a certain amount of coal lost in working. The Royal Commission on Coal Supplies of 1903 took the tonnage to be 1500 per foot per acre, but this was subject to deductions for waste in working, and for pillars, barriers, etc.

Taking the present annual output of coal from the United Kingdom to be 230,000,000 tons, and assuming the average thickness of the seams from which it is produced to be 4 feet, and the yield per foot per acre, 1500 tons, the number of acres worked out per annum amounts to the enormous total of 38,333; that is, to nearly 60 square miles.

Produce of Inclined Seams.—Where a seam is inclined the number of tons under an acre measured on the plan is, of course, more than if the seam were level. The increase on the tonnage depends upon the inclination of the seam, and the tonnage of a seam when level bears the same proportion to the tonnage of the seam when inclined, as the base of a right-angled triangle does to the hypothenuse.

In Fig. 24 is shown a seam dipping at the rate of 1 in 3; the length of the hypothenuse AC, which represents the inclined seam, $= \sqrt{AB^2 + BC^2}$, so that as AB = 1 and BC = 3, the length of AC is $\sqrt{1^2 + 3^2} = \sqrt{10} = 3{\cdot}162$. Then the tonnage per acre (on plan) in the level area bears the same proportion to the tonnage when inclined as 3 bears to 3·162. Hence, if the yield when level is 1500 tons per foot per acre, the yield, when inclined at an angle of 1 in 3, is $\frac{1500 \times 3{\cdot}162}{3} = 1581$ tons.

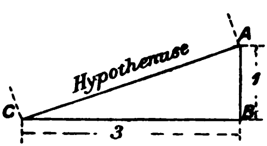


FIG. 24.

A simpler method, to those familiar with trigonometry, is to multiply the tonnage calculated as for a level seam by the secant of the angle of dip.

Modes of Occurrence of Coal.—Coal is very widely distributed among the stratified rocks, being found in almost every division from the Cambrian upwards.

In Great Britain coal is mined almost entirely from the Carboniferous formation, being obtained from the middle or true coal-measures, except in Scotland, where there are large collieries working coal from strata corresponding to the Carboniferous limestone. Coal is, or has been, worked to a limited extent, in England, from the newer measures, and there are very large deposits of such coal abroad.

Coal-seams differ very greatly in character, their main differences being as follows :—

Thickness.—The thickest seam in England is the South Staffordshire thick coal, which attains in some places a thickness of about 33 feet of clean coal; whilst in other districts seams of under one foot in thickness are being profitably worked. In districts where no thick seams exist, it is possible to work very thin seams to a profit; but, generally speaking, thin seams cannot compete with thick ones in the same district.

Inclination.—Seams occur at all inclinations, from vertical to horizontal. The flatter seams are the more cheaply wrought, and a much larger output is possible from a flat seam than from a steep one. In some districts the measures have no regular dip, but are undulating; this adds to the cost of haulage, and greatly increases the cost of dealing with water if any be present.

Purity.—Coal is subject to two classes of impurities—one consists of shale, iron pyrites, and earthy matters, which can be washed out; and the other class consists of chemical impurities, chiefly sulphur and phosphorus, which cannot be removed.

The cleaner and better seams are becoming exhausted, and, in order to make the dirtier seams marketable, very elaborate and costly cleaning plants are being introduced.

A coal-seam usually consists of layers or bands; these may all be coal of various qualities, or one or more of these bands may consist of dirt. The following are typical examples, taken from various districts :—

Barnsley Bed.

(South Yorks.)

	Coal.	Dirt.
	ft. ins.	ins.
"Bags"	1 0	—
Top softs	2 1	—
Clay dirt	—	2
Clay seam	0 9	—
Hards	2 6	—
Bottom softs	1 11	—
Total	8 3	2

Black Shale.

(Derbyshire.)

	Coal.	Dirt.
	ft. ins.	ins.
Top coal	1 5	—
Top dirt	—	4
Tinkers (inferior)	0 9	—
Bottom dirt	—	3
Bottom coal	1 6	—
Total	3 8	7

Hutton Coal.

(Durham.)

	Coal.	Dirt.
	ft. ins.	ins.
Top coal	0 8	—
Bottom coal	4 10	—
Dirt band	—	2
Inferior coal (not worked)	0 5	—
Total	5 11	2

SEVEN-FEET MINE.

(West Lancashire.)

	Coal.	Dirt.
	ft. ins.	ins.
Top coal (inferior)	2 0	—
Dirt band	—	10
Bottom coal	4 9	—
Total	6 9	10

MAIN COAL.

(South Derbyshire.)

	Coal.	Dirt.
	ft. ins.	ft. ins.
Rider coal	3 0	—
Clunch	—	1 0
Lower rider coal	0 6	—
Dirt	—	1 0
Over coal (inferior)	6 0	—
Dirt	—	0 6
Tops	0 7	—
Best hards	1 3	—
Middle dice	2 3	—
Spires	2 9	—
Grounds (inferior)	1 6	—
Total	17 10	2 6

The over coal is left for a roof, the workable portion of this seam being that portion between the grounds and over coal.

Hardness.—Seams differ very greatly in this respect, some being exceedingly hard, and others of a tender nature. The hard coals are usually more expensive to work, but naturally make much less small. Generally speaking, the deep seams make more small than the shallow ones, and coal mined from very deep seams is apt to decrepitate on reaching the surface.

The method of working has a great influence on the amount of small coal, but some seams invariably make much more small than others.

Cleavage.—The "cleavage," "bord," or "face" lines are vertical joints in the coal; they occur every few inches, and their effect is to divide the coal-seam into vertical layers or slabs.

Bord lines are very strongly marked in some districts, and may run with perfect regularity for hundreds of yards; they are generally more strongly marked where the bord line is parallel to the strike of the seam. When the bord lines are very distinct they have a great influence on the method of working the coal, as the coal is got much more easily "on bord," that is, when the working face is parallel to the cleavage planes. In house-coal pits, the coal is often worked "on end," that is, at right angles to the bord; or "on the cross," that is, making an angle with bord and end, as by working in this manner much more round coal is obtained. In some districts the bord lines are very indistinct, and are disregarded in working the coal.

Varieties of Coal.—Coal consists of carbon, volatile matters, and ash. Several classifications have been suggested, but the most convenient depends upon the relative proportions of carbon each variety contains. The different varieties of coal gradually merge one into the other, there being no well-defined line between any two of them, so that there is considerable variation in the chemical composition of the different coals belonging to the same class.

Peat.—This is considered to be the first step in the change of vegetable matter into coal; it is a brown fibrous material, very light and friable. It is found in peat bogs, sometimes up to 60 feet in thickness, and occurs in many places, notably, Ireland, Scotland, Lancashire, Lincolnshire, etc. It is used locally as fuel, but owing to its bulk and to the large amount of water it contains, it has never become a commercial product as a fuel, though efforts have been made to find a market for it, by drying it and compressing it into blocks. The average chemical composition of peat is—

Carbon.	Hydrogen.	Oxygen and nitrogen.
55 to 65 per cent.	5 to 7 per cent.	30 to 40 per cent.

Lignite, or Brown Coal.—Lignite is the next link between vegetable matter and true coal. It is of a woody fibrous texture, and is brown, or brownish-black in colour; it contains much water, and burns with much smoke and smell. Lignite has been worked on a small scale in Devonshire, from a seam found in Lower Tertiary strata, and is largely worked in several places abroad. The chemical composition of lignite is—

Carbon.	Hydrogen.	Oxygen and nitrogen.
65 to 75 per cent.	5 to $7\frac{1}{2}$ per cent.	15 to 30 per cent.

Lignite gives off little heat, and contains a large amount of ash, sometimes as much as 30 per cent.

Bituminous Coal.—Nearly the whole of the coal worked in Great Britain belongs to this variety, which has many subdivisions, such as house, gas, and steam coal; cannel or parrot coal, etc. Several of these varieties are often found in the same seam, each variety forming a distinct band; thus, in the section of the Barnsley bed given earlier in this chapter, the "hards" are best steam coal, whilst the "softs" are often sold as a gas or house coal.

Caking coals fuse or cake together when burning, and appear to be more highly bituminous than the non-caking or free-burning variety, though there may be little difference in their chemical composition. Caking coals make good coke, free-burning coals do not.

Cannel.—This is a dull hard variety of coal, showing no signs of cleat or lamination, it is found chiefly in the Lancashire and Scotch coal-fields.

Sometimes a seam is cannel throughout; at other times a portion only of the seam is cannel; and seams which are composed wholly of cannel in one place may consist entirely of ordinary coal a short distance away.

Cannel is very rich in hydrogen, and is of great value as a gas coal. A ton of good cannel coal should yield from 14,000 to 16,000 cubic feet of gas of about 40 candle-power.

The average chemical composition of bituminous coal is—

Carbon.	Hydrogen.	Oxygen and nitrogen.
About 85 per cent.	5 to $5\frac{1}{2}$ per cent.	8 to 12 per cent.

Anthracite.—This variety of coal is not extensively worked in Great Britain, though large quantities exist in South Wales.

In Pennsylvania it is worked on a very large scale, and is used as a house and steam coal. In England it is mainly used for malting and other purposes which require a smokeless fuel.

Anthracite is very hard and black, it does not soil the fingers, and breaks up into cubes.

It is supposed to be the last stage in the formation of coal, and is no doubt bituminous coal altered by heat and pressure. In Wales it frequently happens that seams which are anthracite in one place gradually change into ordinary steam coal in another. Anthracite burns without smoke and gives off great heat. It requires a strong blast for combustion, and has to be broken up into small cubes before use.

The chemical composition of anthracite is—

Carbon.	Hydrogen.	Oxygen and nitrogen.
90 to 95 per cent.	$2\frac{1}{2}$ to 5 per cent.	$2\frac{1}{2}$ to 5 per cent.

It will be noticed that the gradual conversion of vegetable matter into coal is accompanied by a loss of the volatile matters, leading to an increased percentage of carbon. This is clearly shown by the following table, which is taken from Andre's "Mining Engineering" (E. and F. N. Spon):—

	Specific gravity.	Carbon.	Hydrogen.	Oxygen and nitrogen.
Wood	0·91	49·00	6·25	44·75
Peat	0·99	59·30	6·52	34·18
Lignite	1·25	72·37	5·18	23·45
Cannel coal	1·27	80·07	5·53	13·50
Bituminous coal	1·30	86·17	5·21	8·62
Semi-bituminous coal	1·37	91·00	4·75	8·62
Anthracite coal	1·50	92·50	3·75	3·75

Calorific Value of Coal.—The heating properties of different kinds of coal depend mainly upon the amount of carbon they contain. The oxygen present in coal is usually found combined with a portion of the hydrogen in the shape of water, so the oxygen has no heating value, neither has the hydrogen with which it is combined. In gas coals there is always an excess of hydrogen present; this is known as "disposable hydrogen," and it is only this disposable portion of the hydrogen which is available for heating purposes. The heating power of coal is measured in "heat units," the British heat unit is the amount of heat required to raise 1 lb. of water by 1° Fahr. A good steam coal should give from 14,000 to 15,000 units per pound.

Uses of Coal.—House coal should take fire readily, be clean and free from white ash, leave no clinker, and give a bright fire with little smoke. It should be fairly hard, so as to make little slack. Gas coal should yield at least 11,000 cubic feet of gas of 16 or 17 candle-power, and should give a good gas coke. Cannel coal gives gas of a much higher candle-power, but the coke produced from it is of little value. Steam coal should be hard and hot-burning, be fairly free from sulphur, and not too high in ash. The free-burning varieties are preferable to the caking coals, as the latter hinder the draught by clogging the firebars, and put more work on to the stokers. Coal for metallurgical purposes must be very free from sulphur and phosphorus, and should give off great heat without much smoke or flame.

British Coal-fields.

The following is a brief description of the most important British coal-fields.

The Northern Coal-field.—This coal-field extends from the river Coquet nearly down to the river Tees, the length from north to south being about 50 miles, and the width

varying between 5 and 30 miles. It lies partly in the county of Durham and partly in Northumberland. The strike of the measures is roughly north and south, and the dip is towards the east.

The coal-measures outcrop towards the west, where they are bounded by the millstone grit, to the east dip under the sea, towards the south are overlain by Permian rocks. The only extension of this coal-field is towards the south and under the sea, and mining is being vigorously prosecuted in these directions, both by the old collieries of Seaham, Ryhope, Monk Wearmouth, etc., and by the new winnings at Easington and Horden.

This coal-field is not much troubled by faults, and the seams are found regular in thickness and lying at a light inclination. Several large Whin dykes traverse the district in a south-easterly direction.

The Cumberland Coal-field.—This coal-field lies along the Cumberland coast, extending from a little below Whitehaven up to Wigton, and having a length of about 25 miles and a width of about 5. The coal-measures dip towards the west under the Irish Sea, and are bounded on the east by the millstone grit. Towards the north and south they are overlain by the Permian rocks, under which extensions may be expected.

The principal seams are the Bannock Band, Main Band, Ten Quarters, and Metal Band. The Whitehaven Collieries have already worked the main band for a distance of over three miles under the sea. The Workington Colliery was also working the main band under the sea until the year 1837, when, by the injudicious removal of pillars, the water broke in with disastrous results. This coal-field is much cut up by faults.

The Midland Coal-field.—This great coal-field extends from below Nottingham to Leeds, having a length of about 65 miles, and lying in the counties of Nottingham, Derby, and

York. The strike of the beds is about north and south, and the measures dip gently towards the east, where they are overlain by the rocks of the Permian, Trias, and Lias formations. On the west and north the coal-seams crop out one after the other, and are bounded by the millstone grit, and on the south they rise and crop out under the Permian rocks. The only direction in which extension can be looked for is towards the east, and there great developments may be expected, and indeed are taking place. The coal-measures have been proved to extend for 10 miles to the east of Doncaster by the South Carr borehole, which struck the Barnsley bed at a depth of 1050 yards. How far the coal-measures actually extend to the east is not known; some authorities believe that they come to an end by rising to the east and outcropping under the Permian, whilst others hold that they extend as far as the North Sea. They may terminate either by thinning out or by being cut off by the thickening of the Permian measures.

The principal seam is the Barnsley bed, which is found at its best in the neighbourhood of the town from which it takes its name. In the South Yorkshire district this seam has a thickness of from 7 to 10 feet. In Derbyshire it is known as the Top hard coal, and is of excellent quality. It has been worked continuously from Nottingham to Barnsley. Towards the north the Barnsley bed splits up and takes the name of Warren House, and is only a second-class seam. Other important seams are the Parkgate and Silkstone, the former being the Deep hard and the latter the Blackshale of Derbyshire.

The lower coal-measures also contain thin but valuable seams of coal, accompanied by excellent beds of ironstone and fire-clay.

Taken as a whole, this coal-field is not greatly troubled by faults, and no intrusions of igneous rock have been met with. The inclination of the beds is generally moderate.

The Lancashire Coal-field.—This coal-field is separated from the Midland coal-field by the Pennine chain anticlinal,

and at one time the two were connected; the Ganister and Silkstone seams of Yorkshire probably representing the Mountain and Arley mines of Lancashire. On the north and east the Lancashire coal-field is bounded by the millstone grit, but to the south and west the coal-measures are overlain by Permian rocks, which have not been thoroughly explored. There are many valuable seams of coal, at some collieries as many as ten being worked. In the St. Helen's district the Ravenhead, Rushey Park, and St. Helen's Main delfs are most sought after; whilst in the neighbourhood of Wigan, the Wigan and Orrell mines have been extensively wrought, and contain excellent cannel.

The coal-field is much intersected by faults, and the inclination of the measures is, as a rule, very considerable.

The North Wales Coal-fields.—These coal-fields occur in the counties of Denbigh and Flint. On the west they are bounded by the millstone grit and mountain limestone, but dip under the Permian rocks to the east, in which direction further extension is probable.

On the Flintshire coast the coal-measures extend under the Dee, and are worked on the Wirrall peninsula. The chief coal-seams in the Ruabon district are the Main, Yard and Bench, and Quaker, and in Wirrall the Six-feet, Five-feet, and Seven-feet.

The North Staffordshire Coal-field.—This coal-field underlies the Pottery district of North Staffordshire, the chief towns upon it being Stoke and Hanley. It is bounded on the east by the lower Carboniferous rocks, but the coal-measures extend to the west and south under the Permian rocks.

The coal-field is extremely rich, but somewhat irregular, the seams being frequently very highly inclined (when they are known as "rearers"), and in some cases even vertical.

The Leicestershire Coal-field.—This little coal-field is situated around the town of Ashby-de-la-Zouch, and probably

at one time formed part of the Midland coal-field, being separated by an anticlinal having an axis running from west to east. On the north-west it is bounded by lower Carboniferous strata, but on the west and south it is covered by the new red sandstone under which it extends. Six or eight seams have been vigorously worked, the most valuable being the Main, Eureka, and Stanhope seams in the Moria district, and the Main and the Roaster Main in the Coalville area.

The Warwickshire Coal-field.—The exposed portion of this coal-field extends from Tamworth on the north to Bedworth on the south. It is almost entirely surrounded by Permian strata, and recent winnings appear to point to the coal-measures extending under these Permian rocks up to the South Staffordshire coal-field, a distance of about 12 miles.

The principal seams are the Slate, Ell, Rider, and Two-yard coals; and at some collieries these are found separated by only a few feet of strata, and are worked together.

The South Staffordshire Coal-field.—The exposed portion of this coal-field extends from Cannock to the Clent Hills, but as it is bounded on all sides by the new red sandstone, it is probable that great extensions will be made. Collieries which have been sunk through the new red measures on the east and west, afford evidence which renders it probable that the coal-field joins up to the Warwickshire and Shropshire coal-fields. The chief feature is the presence of the Ten-yard coal, which is found in the southern portion of the area. This seam is at its best in the neighbourhood of Dudley, where it is 33 feet in thickness. Igneous dykes occur in places.

The Shropshire Coal-field.—The Shrewsbury, Coalbrookdale, and Forest of Wyre coal-fields, extend in a broken line from Shrewsbury to the river Teme; towards the west the coal-measures terminate against Pre-carboniferous rocks, but extensions may take place under the new red sandstone,

which forms the northern and eastern boundary of the exposed coal-field.

The Bristol and Somerset Coal-field.—This coal-field lies to the north-east of the town of Bristol. The exposed area is small, but a much larger area is covered by newer formations. One of the chief features of this coal-field is the thinness of the seams, some which are being worked are only 10 inches in thickness.

The Forest of Dean Coal-field.—This little coal-field has an area of 34 square miles, and is situated around the towns of Coleford and Cinderford, in Gloucestershire. It is in the form of a perfect basin, the seams dipping to the centre from all sides. The seams are thin, and the most valuable are rapidly becoming exhausted.

The South Wales Coal-field.—This important coal-field lies in the form of a basin, its length being about 53 miles from east to west, and its width from north to south about 16. A large anticlinal traverses almost the whole length of the basin, bringing the deepest seams to a workable depth at its crest, but at the lowest parts of the basin the deeper seams lie beyond the reach of present methods of mining. Towards the south-east the seams are bituminous, but they gradually change their character until they become anthracite in the north-west. The coal-measures are divided into an upper and lower series by a great thickness of sandstones, known as the Pennant grit.

The Scotch Coal-fields.—These lie in a long strip extending from the Firth of Forth and the Firth of Clyde, the coal-bearing area being divided into the Midlothian, Clyde, Fifeshire, Ayrshire, and other basins. The coal-measures are divided into the upper and lower series, which are divided by the Millstone grit. The lower series are in the Carboniferous limestone, and contain valuable beds of coal and cannel. Below them are the Calciferous sandstones, which

contain the oil shales, largely worked in the neighbourhood of Edinburgh.

The Irish Coal-fields.—These are small and unimportant. They occur in Antrim, Leitrim, Leinster, Tipperary, and West Munster.

The Dover Coal-field.—Little is at present known of the character and extent of this coal-field. Several boreholes have been put down, and shafts are now in process of sinking (1904). The boreholes proved several seams of coal, the principal one, 4 feet in thickness, being found at a depth of 2225 feet. It is supposed that a line of detached coal-basins extends across England, from the Bristol coal-field to Dover, and probably runs under the sea and joins up to the Belgian coal-fields.

The relative importance of the various coal-fields may be estimated from the following table, which gives the output for the year 1902 :—

	Tons.	Percentage of the total output.
1. Scotch coal-fields	34,115,309	15·0
2. Northern ,, ...	46,427,487	20·4
3. Yorkshire, etc., coal-fields	52,136,750	23·0
4. Lancashire and Cheshire coal-fields ...	24,879,391	11·0
5. Midland coal-fields (Staffordshire, Shropshire, etc.)	20,264,442	8·9
6. Small detached coal-fields	4,674,054	2·1
7. North Wales ,, ...	3,173,118	1·4
8. South Wales ,, ...	41,305,583	18·2
9. Irish coal-fields	108,737	—

CHAPTER IV.

SEARCH FOR COAL.

Preliminary Operations.—Before making any detailed search for coal in any particular locality, it is first necessary to ascertain the geological period to which the rocks forming the surface of that district belong. If the rocks are found to belong to a period older than the coal-measures, further search will be useless, as no workable coal is likely to be found; if the surface rocks are much newer than the Carboniferous, coal may be present, but probably at a great depth; but if the coal-measures themselves are found on the surface, a detailed examination for outcrops should be made. If no geological map of the district exists, the explorer must make one for himself, and to do this will require considerable geological knowledge. An ordinary map of the locality must first be procured, or, if necessary, made; and upon this map must be marked the nature and dip of the rock beds wherever they are exposed. Sections of the beds may usually be seen on the banks of streams or rivers, in railway or other cuttings, in quarries and wells, and sometimes on the hillsides; by examining all these, and by carefully recording the information upon the map, the general geological features of the district will be gradually determined.

Fossils are of the greatest assistance in determining the geological period to which rock beds belong, and any that are found should be carefully preserved, as their accumulative evidence may definitely determine the system of the rocks in which they were found.

A seam of coal is usually thin, and always of inferior quality at its outcrop. To test its true character, trial headings should be driven into the seam at intervals along the outcrop; these will not only prove the seam itself, but will also test its continuity and reveal any dip faults which may be present. If more convenient, trial shafts may be substituted for the headings.

Before a decision can be arrived at as to whether or not a coal-seam is likely to be workable at a profit, many factors have to be noted and considered, chief among which are the following:—

Thickness.—The minimum thickness of coal which can be worked at a profit is almost entirely a matter of locality. Usually it is impossible to work much thinner seams than are worked at neighbouring collieries. In some districts seams of 18 inches and under in thickness can be profitably worked, whereas in other districts this would be quite impossible.

Quality.—This, too, is to some extent a matter of locality. A coal-seam, to be worked at a profit, must be as good as those with which it will have to compete; otherwise there will be no sale for it when trade is depressed, except at ruinous prices.

Nature of Roof and Floor.—Rocks, strong binds, or coals make the best roofs, and the nature of the roof has the greatest influence on the cost of working a coal-seam. The floor should not be too soft, or it will "heave," and cause the roads to be very expensive to maintain. The presence of water in roof or floor is almost fatal to economical working, when they consist of "binds," or "clunches."

Quantity of Water present.—It is generally very difficult to estimate the quantity of water which may be made in the workings. The expense of dealing with water does not so much depend upon the quantity present, as upon the difficulties which may be met with in concentrating it, and the general conditions of each mine.

Faults.—In an unproved coal-field, faults are often quite unexpected until they are actually struck in the workings.

They may be a source of enormous trouble and expense; they not only entail the cost of crossing them, but they disarrange the haulage and upset the whole scheme of the workings. Faults may occasionally be of advantage, by acting as a barrier and keeping back water; they frequently form a dividing line between two collieries.

Available Area.—The capital expended upon a modern colliery is so great that very large areas are now the rule. In the Midland district the newer royalties are seldom less than four or five thousand acres. The royalty rent paid per acre averages about £30 per foot thick, which works out to about 5*d.* per ton on the output.

Labour.—Most new collieries have to provide dwelling-houses for their workmen; this increases the capital required, but money spent on cottages yields good interest.

Markets.—The railway rates to the largest markets have, of course, a very important bearing on the profits; if possible, a large colliery should be in communication with at least two railways, as the competition between them leads to much better facilities for traffic. In the neighbourhood of large towns the land sales may be large and remunerative. Canals also afford a means of transport to many important centres.

Boreholes.—When a coal-field is overlain by rocks of a newer period, as is the case with most of our undeveloped areas, surface explorations are of little value, and boring has to be resorted to.

It frequently happens that the coal-field is proved on one or more sides by other collieries, and the borehole is only required to complete the information thus gained. When this is the case one borehole only may be required.

At other times it is desired to prove lower seams under an area from which the upper seams have been worked; this is best done by putting down a borehole from the bottom of an existing shaft. If the district is quite unexplored, several boreholes are required to prove a large area, and the sites have to be very carefully chosen in order to obtain the most complete

and reliable information. A borehole should show the depth, thickness, and quality of the seams passed through.

A borehole only shows the true thickness of the seam passed through when the seam happens to be level; when it is inclined, the thickness is arrived at by multiplying the distance bored through by the cosine of the angle of dip. The following table gives the true thickness of a seam per foot bored through when the seam lies at various inclinations:—

Thickness of coal bored through.	Dip of measures in degrees.	True thickness of seam.
1 foot	5°	0·996 feet
,,	10°	0·985 ,,
,,	15°	0·966 ,,
,,	20°	0·940 ,,
,,	25°	0·906 ,,
,,	30°	0·866 ,,
,,	35°	0·819 ,,
,,	40°	0·766 ,,

If a borehole passed through a seam dipping 20°, the thickness of coal in the borehole being 4′ 3″, the true thickness would be $4{\cdot}25 \times 0{\cdot}940 = 3' \, 11''{\cdot}94$. Of course, such calculations are only correct when the borehole is vertical.

When boreholes are required to prove the thickness, rate, and direction of dip and depth of a seam of coal in a totally unproved district, at least three are necessary; they might be set out as in Fig. 25. Their distance apart would depend entirely upon local conditions.

To find the direction and rate of dip of a coal-seam from information obtained by three boreholes, proceed as follows:—

First, prepare a plan of the three boreholes, A, B, and C, Fig. 25, in their correct positions to any convenient scale. In the present case, A is 130, B 205, and C 170 yards deep; these may not be the actual, but are the corrected depths after making allowance for the variations in surface levels. The coal-seam is 75 yards deeper at B than at A, and as the

horizontal distance AB is 300 yards, the inclination of the measures from A to B is 75 in 300, or 1 in 4.

The seam at C is 40 yards deeper than at A, and the distance AC is 360 yards, hence the dip from A to C is 40 in 360, or 1 in 9.

Now, the dip of AB is 1 in 4, so that at a point 4 yards from A measured along AB the seam will be exactly 1 yard deeper than at A; and the dip along AC is 1 in 9, so that 9 yards from A measured along AC the seam will again be exactly 1 yard deeper than at A; hence it follows that a point 4 yards from A measured along AB is exactly level with a point 9 yards from A measured along AC. To put this level

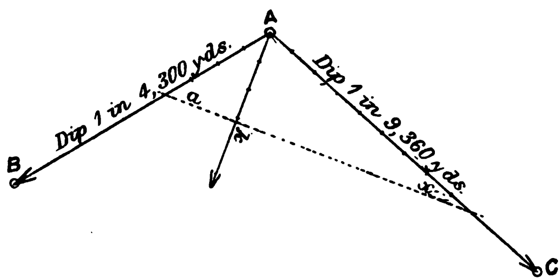


FIG. 25.—Method of finding true dip from three boreholes.

line upon the plan, mark off on AB any four equal parts and nine similar parts along AC; join these points and the line between them (*ax* on sketch) will be the level line of the seam, and a line drawn from A at right angles to it (A*k*, Fig. 25) will show the line of full dip. To get the rate of inclination, measure the line A*k*, using the same scale: in the figure this line measures 3·2, and *ax* is exactly 1 below A, so that at full dip the seam gained 1 vertically in 3·2 horizontally, and its dip is 1 in 3·2, or about 17°. Having thus found the rate of inclination, the true thickness can be calculated from the thickness bored through, in the manner previously explained. The direction of full dip can be read off the plan by means of a protractor.

Of course, if any faults exist between these boreholes the

deductions will be incorrect; this is the weak point in the reliability of the results obtained by boring.

Boreholes may be put down by one of two systems—the *percussive* and the *rotary;* the former is the older of the two, but for important work in connection with mining is now almost entirely superseded by the latter.

The Percussive System of Boring.—This system of boring is only suitable for unimportant holes of moderate depth, as for very deep holes it cannot compare with the rotary system, either as regards cost or speed. Moreover, the results cannot be considered satisfactory, because the samples from the seams bored through are cut up into such small fragments that the evidence as to their quality is never very reliable.

The process of putting down a borehole by the ordinary percussive system is as follows :—

A *guide pipe* is first driven vertically into the ground; it consists of a wooden or iron pipe, of the same diameter as the hole is to be started, and from 6 to 9 feet in length. It is fitted at its upper end with iron covers or shutters, which have a square hole in them to allow the passage of the rods. The covers are required to prevent anything from falling down the hole, and can be moved clear of the hole when required.

Exactly over the guide pipe *a derrick* or *head-gear* is erected. In its simplest form this consists of three poles, forming a triangle at their base, and coming together at the top. The derrick carries a pulley, as shown in Fig. 26, and a windlass is fitted across two of the legs. A piece of hoop iron is nailed around the legs near the top, to form a guide for the rods when they are raised out of the hole.

The rods are sometimes worked by means of a *lever* or *brake,* in which case they are suspended to its short arm, and the power applied through the other. For shallow holes a spring-pole is frequently employed; it consists of a larch pole about 30 feet in length, arranged as shown in Fig. 26. When the end of the pole is depressed, the blow is struck, and the rods are raised by its elasticity.

The hole is bored by *chisels*; the chisels are attached to the *bore rods*, which are screwed to the *bracehead*, and hung from the *lever* or *spring-pole* through a *stirrup*.

The *chisels* (*a* and *b*, Fig. 27) are about 18 inches long, and made of best tool steel. The flat chisel (*a*) is the most common, and is suitable for the strata usually met with in the coal-measures; for exceptionally hard ground chisels shaped in the form of a cross are employed. The ends of the chisels are fitted with bosses and screws, by means of which they are attached to the rods.

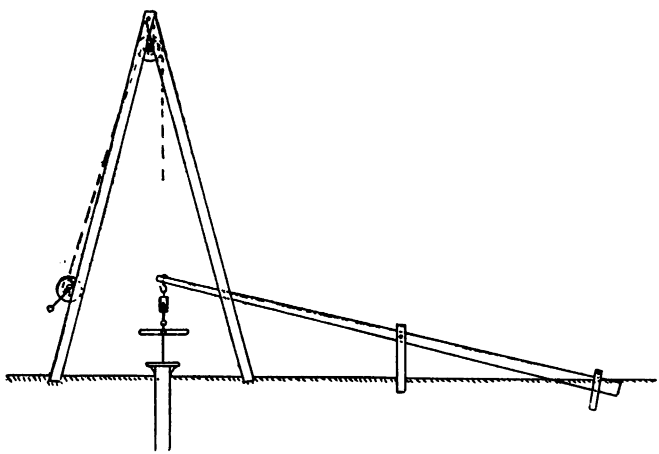


FIG. 26.—Surface arrangements for percussive boring.

The *rods* (*c*, Fig. 27) are of best wrought iron, and vary in size from $\frac{3}{4}$ to $1\frac{1}{8}$ inch square, 1 inch square being the most common; they are made in lengths of up to 18 or 20 feet. The ends are provided with bosses of about double the diameter of the rods themselves, and one end of each rod has a male screw, and the other a female screw.

Wooden rods are also employed, but are not common in this country.

The *bracehead* (*d*, Fig. 27) consists of four wooden arms, each about 18 inches in length, fitted into an iron socket. At

the top of the bracehead is a swivel, and at the bottom a short length of rod terminating in a screw.

The *stirrup* (*e*, Fig. 27) consists of a bridle of iron and long screw; its position is between the bracehead and lever, or spring-pole.

In boring a hole, the tools are hung from the lever or spring-pole in the following order: stirrup, bracehead, rods,

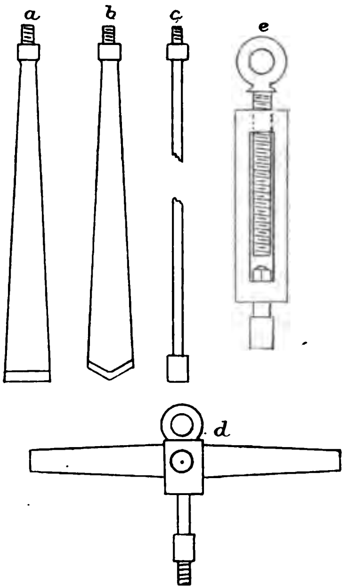


FIG. 27.—Boring tools.

and chisel. Two or four men each take an arm of the bracehead and lift the rods up, allowing them to fall sharply; as they do this the men walk slowly round in a circle, so that at each blow the chisel is moved through a small angle, and strikes the bottom of the hole in a fresh place; this prevents the chisel from jamming, and keeps the hole circular.

As the hole gets gradually deeper the rods are automatically lengthened by the lowering of the screw in the stirrup, and

when this has reached its limit, a short rod is added to the top of the other rods, and the screw in the stirrup is run back. After a few inches have been bored, the bottom of the hole gets full of *débris*, which has to be removed. To do this, the bracehead is unscrewed, and the hook at the end of the windlass rope (*a*, Fig. 28) is slipped under the boss of the top rod, the windlass is turned, and the rods raised slowly out of the hole to the height of the derrick. A fork (*b*, Fig. 28) is then slipped underneath the lowest boss above the guide pipe, to prevent the rods from descending the hole, and the rods above are unscrewed by a spanner; this process is repeated until all the rods have been raised from the hole. A shell or sludger (*c*, Fig. 28) is then screwed on to the end of the rods and lowered down to the bottom of the hole.

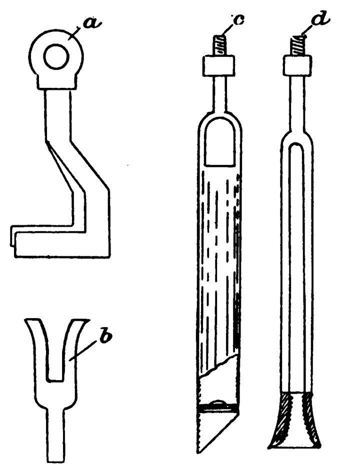


FIG. 28.—Boring tools.

The sludger consists of an iron pipe, closed at the bottom by a valve opening upwards, but open at the top. It is allowed to strike the bottom of the hole violently several times; this forces the *débris* into the pipe, where it is retained by the valve. When all the *débris* has been collected by the sludger, it is raised to the surface and emptied; the chisel is again screwed on to the rods, sent down the hole, and the boring continued. These operations succeed each other until the borehole is completed.

The nature of the strata passed through is ascertained by examining the contents of the sludger. The master borer is able to tell the principal variations of the strata by noticing the jar upon the rods, which he marks whenever he believes

that a new sort of stratum has been reached. It is not possible to get a very accurate section of the strata by this method of boring, as small bands may be passed through without being detected, although the thickness of any beds of coal can be almost exactly determined by a careful and experienced borer.

When the borehole is very deep, the time absorbed in lifting and lowering the rods in order to clean the hole and change the chisels is very great; in some cases only an hour or two out of the twenty-four have been occupied in the actual process of boring, the remainder being taken up by raising and lowering the rods. In deep holes accidents may occur very frequently, the most common being the fracture of the rods. Various tools are used for catching and lifting broken rods; if the fracture is a clean one, the *bell-box* (*d*, Fig. 28) may be employed to catch the broken end. This is an arrangement with a bell-shaped end made of hard steel, having a thread cut inside the bell. The apparatus is screwed on to the rods, lowered down the hole, and moved about until the broken end of the rod is caught in the bell. The rods to which the bell-box is attached are then turned slowly round, until the bell attaches itself firmly to the fractured rod end by cutting a screw thread on it, and the whole lifted together to the surface. When the fractured end is ragged and bent, a special tool may have to be made to clutch it. An impression of the fractured end may be obtained by lowering a lump of soft wax on to the fracture, and keeping it there until it hardens; a special tool can then be made suitable for the particular condition shown by the impression in the wax. Fragments of chisels which have become broken have sometimes been drawn from the hole by means of magnets lowered down to the bottom.

Lining Boreholes.—Boreholes have usually to be lined to prevent their sides from falling in. This is done by forcing wrought-iron tubes down the hole in the following manner: The tube is first driven as far as possible down the hole by striking it with a hammer. When it can be driven no further by the hammer, a length of rods is let down the hole through

the tube, the top rod being gripped by a strong clamp, which prevents it from slipping down the hole. The rods are then raised a few feet by the windlass and dropped, when the clamp comes into violent contact with the top of the tubes, and forces them down. The tubes have screwed joints, and should be put down in as long lengths as possible, for each succeeding length of tubes will be smaller than the previous one, as it will have to pass through it. This reduces the size of a borehole as it gets deeper, and many holes have been so reduced that they have had to be abandoned before reaching the required depth.

Boring from Colliery Workings.—When boreholes are put down from the workings, a small shaft should be sunk for a few yards, to give room for the rods to be raised and unscrewed. Rubber bands may be substituted for the lever or spring-pole; they are made out of rubber about 1 inch square, extra bands being added as the hole gets deeper and the rods heavier.

When deep holes are bored by the percussive method, the rods should be raised and dropped by mechanical power instead of by hand. This may be done by means of pins which project from a revolving wheel driven by a steam-engine; these pins depress one end of the lever, to the other end of which the rods are attached, and so raise the rods and let them fall. Another method is to attach a hemp rope to the rods; the rope is taken over a pulley in the head-gear, and makes two or three turns round a revolving drum. When tension is put on to the loose end of the rope it binds on the drum, and the rods are raised, and when the rope is released it slips on the drum, allowing the rods to fall.

Boring by Ropes.—Many of the disadvantages of the ordinary method of percussive boring are overcome by the use of a rope instead of rods; in this way holes can be put down much more quickly, but it is difficult to keep them vertical, and not easy to get an accurate section of the strata.

The most elaborate method of rope boring is Mather and Platt's arrangement; an outline of the necessary surface arrangements is shown in Fig. 29.

In Fig 29, *a* is a vertical cylinder having a pulley, *b*, fitted into a fork at the top of its piston-rod. The flat rope to which the boring tool is attached passes over *b*, through the clamp *c*, and round the guide pulley *e*, on to the drum *d*. When boring is in progress, the clamp *c* is tightened and holds the rope fast; steam is then turned into the cylinder *a*, which works the pulley *b* up and down at the rate of about twenty-five strokes per minute. The effect of this is to raise and drop the rope and the heavy tool which is hung on to its end; this cuts into the rock at the bottom of the borehole.

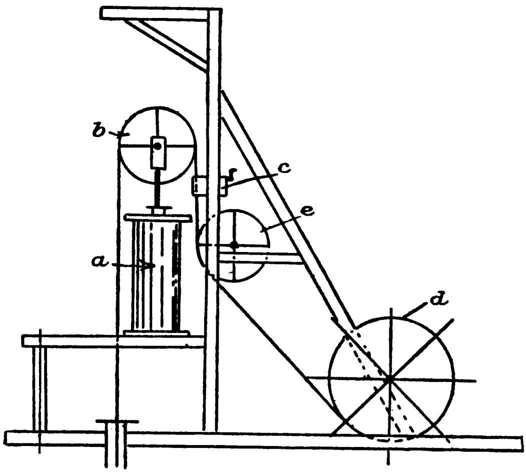


FIG. 29.—Mather and Platt's method of boring.

When the hole has to be cleaned out, steam is shut off from *a*, the clamp *c* is loosened, and the rope is raised by the drum *d*; the sludger is then lowered down at the end of the rope. The boring tool is a heavy iron bar 6 or 8 feet long, having guide blocks at either end. The upper blocks are provided with a rachet arrangement which twists the drill slightly at every stroke, and so forces the chisels to strike every blow at a different place. The bottom block carries a number of chisels, which form the cutting tool.

The sludger is sometimes of special design, or it may be an ordinary "shell" divided into several compartments, each of which has its own valve. This system of boring is not very suitable for proving minerals, as a good section of the strata passed through is not obtainable. Holes can be bored up to 18 or 24 inches in diameter; the cost is said to average about £1 per foot. Round ropes are largely employed for boring in the oil-fields of the United States.

The Rotary System of Boring.—This is now generally acknowledged to be the best system of boring to prove minerals, and all the important boreholes which have been put down of late years to prove coal-seams have been made by one or other of its modifications. The great advantage of this method is that the strata passed through are not cut up into small fragments, but a "core" is extracted. The amount of core actually obtained depends upon the size of the borehole and the hardness of the rock bored through; the larger the hole and the harder the rock, the more perfect is the core.

The holes are bored by abrasion. A ring, made of material harder than the rock which is to be bored through, is rapidly rotated, and is at the same time pressed against the rock. This grinds the rock away where it is rubbed by the ring, leaving a solid column of material, known as a *core*, in its centre, which is broken off and extracted in short lengths.

There are two systems of rotary boring in general use, one known as the Diamond system, and the other as the Davis-calyx.

The Diamond System of Boring.—In this method of boring, the cutting is performed by diamonds of the black variety. The boring-piece is shown in Fig. 30. *a* is a wrought-iron tube about 3 inches in external diameter. It extends to the top of the hole, and the rotary motion is conveyed through it from the engine and gearing on the surface. *a* terminates in a block, *c*; this block forms the division between the sediment tube *b* and the core-box *d*; *b* is an open-topped cylinder rather

less in diameter than the borehole, and *d* is another cylinder of the same size, having the crown *e* screwed on to its lower end.

The boring-crown is a wrought-iron ring, the bottom and both edges of which are set with diamonds. As the crown revolves, these diamonds, which project slightly beyond the metal of the crown, cut away an annular space around a solid central core. Whilst the hole is being bored, water is pumped through the tube *a*; it passes underneath the crown through grooves provided for the purpose, and back to the surface through the borehole. The object of the water is to keep the crown cool and carry away the *débris* made in boring. As the water passes through the restricted area between the boring-piece and the sides of the hole its velocity is very great, so that it carries all the *débris* with it; but as soon as it reaches the unrestricted area above the top of the sediment tube, the velocity decreases, and the *débris* falls and settles in the sediment tube.

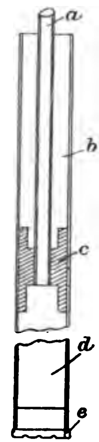


FIG. 30.—Diamond method of boring.

The pressure upon the crown is regulated by means of balance weights, and can be either increased or reduced at will. When the holes are shallow, and the tubes which act as rods light, extra weight is required; but when the holes are very deep, part of the weight of the rods must be counterbalanced.

The cores are broken off by means of a split wedge-shaped ring fixed in the inside of the crown. When a few feet have been bored, the boring is stopped, and the rods are lifted by a steam-winch. As soon as they are raised, the split ring is drawn down an incline and wedges the core tightly in the core-box. It then breaks off, and is drawn to the surface with the boring-piece.

Many very important boreholes have been put down by

this system. At Southcar, in Lincolnshire, a hole was put down to a depth of 3195 feet. Its diameter at the surface was 13 inches, which was gradually reduced, the core at the bottom being about 1¼ inch in diameter. A borehole at Snaith commenced 14 inches in diameter at the top, and finished ½ inch at 1600 feet. Another at Ruddington had a core of about 4½ inches at a depth of 1800 feet.

The Diamond system is suitable for boring through very hard rocks, but the rocks of the coal-measures are not, as a rule, very hard, so that there is no necessity for using so expensive a material as the diamond to pierce them; properly tempered steel will serve the purpose equally well, and in many cases even better.

The Davis-calyx System of Boring.—This method of boring differs little in principle from the Diamond system. Its chief features are the substitution of a steel cutter for the crown set with diamonds. The sediment tube is made very much longer, and is called the "calyx;" its length may be up to 100 feet.

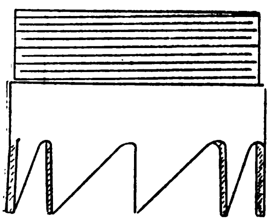


FIG. 31.—Davis-calyx method of boring—cutting tool.

The cutter is shown in Fig. 31. It consists of a cylindrical steel shell, the lower end of which is formed into a series of long sharp teeth. The front edges of these teeth are vertical, and the backs slope at an angle, in order to give the greatest strength in one direction. They are set slightly in and out alternately, like the teeth of a saw. The outside set is to make the hole large enough for the boring tool to pass, and the inside set to dress the core sufficiently small to enable it to pass into the core-box. These teeth have a chipping rather than an abrasive action; they sink into the rock and revolve with a series of jerks, chipping out small pieces of the rock in front of them. The number, size, and shape of the teeth are varied to suit the nature of the ground to be bored through, and the cutters are driven at a

much slower rate than when diamonds are employed. The average speed for a large hole is four or five revolutions per minute.

When the rock is very hard, a plain steel shell, without teeth, is substituted for the cutter described above, steel shot are poured into the cut, and the tool revolved rapidly upon them, thus wearing away an annular groove, as is done by the diamond.

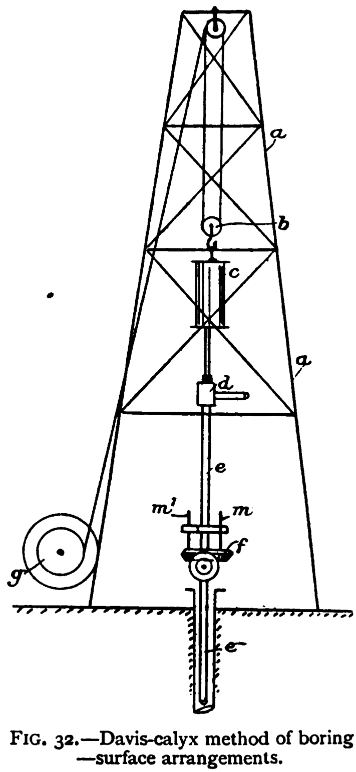


FIG. 32.—Davis-calyx method of boring —surface arrangements.

The surface arrangements are shown in Fig. 32. *a* is a derrick 50 or 60 feet high, from which is suspended a set of blocks, *b*, the block rope being led to the drum *g*. To the top of the rods *e*, which are hollow and about 3 inches in diameter, is attached the swivel and water coupling *d*, to which is connected the pump by a hose. *c* is an hydraulic cylinder, by means of which part of the weight of the rods can be carried, for when the holes are very deep the full weight of the rods could not be safely borne by the cutter. The driving-wheel *f* has a hole in its centre, through which the bore-rod pass. It also carries two arms, *m*, *m'*, which engage with a clamp on the rods, and so force them to revolve with the driving-wheel. By this arrangement the rods are free to slide vertically through the driving-wheel, and so are able to follow the borehole down as it gradually increases in depth.

Both drum and driving-wheel are driven by a small pair of engines, and can be thrown in and out of gear when required by means of clutches. Water is forced down the rods through the hose-pipe by a small pump. About 50 gallons per minute are required. The same water can be used repeatedly, after it has passed through a small settling pond.

Boring operations are conducted as follows: Part of the weight of the rods is held by the block ropes, the exact weight upon the boring tool being regulated by the hydraulic cylinder *c*. The pump is kept going at a regular speed, forcing water down the rods, under the cutting tool, and back up the borehole. The engine is put into gear with the driving-wheel, which rotates the rods through its arms, which catch the clamp on the rods. As the borehole deepens, the clamp, being fixed to the rods, slides down, and has to be moved up every foot or so.

The length of the core-box varies from 8 feet in large holes up to 15 feet in small ones. When the core-box is sufficiently full, the engine and pump are stopped, and the rods raised by means of the drum and block ropes, and unscrewed in sections, the length of the sections which can be unscrewed at once depending upon the height of the derrick. To break the core off, gravel is thrown down the bore-rods, causing the core to bind in the box.

One hundred feet per day have been bored by this process, but this was in chalk, where the drilling would be unusually easy.

Surveying Boreholes.—Boreholes frequently incline very considerably from the vertical. The deflection may be so serious as to render a survey of the boreholes necessary. This may be accomplished in several ways. MacGeorge's clinometer is an instrument sometimes used for this purpose. It consists of a brass cylinder containing liquid gelatine, in which are suspended a plumb-bob and a magnetic needle. The apparatus is let down the borehole, and after a short time the gelatine congeals and holds the needle and plumb-bob in the same position that they occupied when in the hole, so that its exact inclination and bearing can be ascertained.

CHAPTER V.

SINKING.

Site of Shafts.—The choice of the position of the shafts is governed by a variety of circumstances, as both surface and underground conditions have to be taken into account.

In former years, shafts were always sunk to the deep of the coal, because the appliances for pumping and hauling were so crude that it was almost impossible to work dip coal; the shafts too, being shallow, were sunk close together to save the cost of haulage and of keeping long roads in repair. In districts that have been worked out many years ago, long lines of shafts can be traced. The oldest have won a narrow strip of coal alongside the outcrop of the seam; the next line would be deeper and have worked a long narrow strip to the dip of the first, and so on.

The haulage appliances at the present time are so good that dip coal can be worked almost as cheaply as rise coal, except where water is present. Water is much less common in deep pits than in shallow ones, and can be much more cheaply dealt with than formerly by the aid of electric pumps, so that at the present time it is not necessary that the shafts should be at the dip of the royalty.

In selecting the site of a new winning, the following points should be considered:—

(1) There must be ample room for surface works, sidings, coke-ovens, workmen's houses, spoil heaps, etc.

(2) Access to one or more railways must be readily available, and the surface contour should be suited for the necessary

branch lines. The question of landsales and communication with a canal should also be considered.

(3) A supply of good water must be obtainable, as a large quantity will be required for steam raising, coal-washing cottages, etc.

(4) If large faults are known to exist, the position of the shafts may, to some extent, be governed by them.

(5) The shafts should be as nearly as possible in the centre of the royalty.

(6) Surface beds of sand, moss, etc., should, if possible, be avoided, as they may add enormously to the cost of sinking and of the foundations of the various surface erections.

It is obviously very seldom that a site can be chosen to comply with the whole of the above conditions, and frequently the choice is limited to certain areas by the terms of the lease.

Number and Size of the Shafts.—At one time large collieries were worked with one shaft only, which was divided by a brattice for ventilation; but since the accident at Hartley Colliery in 1862, the Coal Mines Regulation Act (§ 16) has stipulated that every mine shall have at least two shafts or outlets in communication with every seam, and not less than 15 yards apart. When several seams are to be worked at once, three or more shafts are sometimes sunk.

Shafts may be either rectangular, polygonal, oval, or circular in form. In England the latter are almost always adopted, being stronger, cheaper to sink, and much superior to the other forms, when the shaft has to be lined with cast-iron tubbing.

In Scotland rectangular shafts are still sunk, and are lined with timber instead of brickwork.

Among the advantages claimed for rectangular shafts, one is that there is little vacant space in them, and another, that they are more easily divided by a brattice, if one is required. The surplus space which occurs in circular shafts is, however, by no means wasted, being required for the passage of the air. Rectangular shafts are frequently employed for staple pits, and

are almost universal in metalliferous mines; there are several oval shafts in South Wales, and many polygonal shafts have been sunk upon the continent. The size of shafts is gradually increasing, owing to the fact that royalties are larger and outputs increasing; most modern shafts are between 15 and 23 feet in diameter. It is not usual to have more than one pair of cages in a shaft, though occasionally two pairs are employed when several seams have to be worked. In many of the old collieries in the county of Durham, two winding engines and two pairs of cages pull from one shaft; examples of this may be seen at Murton, Hetton, and many other places. This practice increases the winding capacity of the shafts, but is hardly to be commended for new places.

In some districts there may still be seen one winding engine drawing from two or three shafts, with one cage in each; this is not a desirable arrangement, as it limits the output from each shaft.

Sinking through Loose Ground.—Strong rocks and shales are frequently found right up to the surface, but in some districts the measures for a considerable depth are composed of very loose, wet ground, such as running sand, gravel, and moss. Special methods have to be adopted for sinking through such material. When the loose ground is not of great depth, it may be got through by means of piles; but when the thickness is very great, sand tubbing or some other system of sinking has to be resorted to.

Sinking by Piles.—Before commencing to sink through loose ground by piling, a borehole must be put down to prove the thickness of the measures which will have to be piled through; this is necessary, because in the ordinary system of piling the size of the shaft is gradually decreased, so that the length of piling required has to be known before commencing operations, in order that the initial diameter may be arrived at.

The surface soil is first removed, and the ground levelled. A curb (*a*, Fig. 33) made of oak, 6 inches square in section, is then laid on the ground in a perfect circle, as shown in Fig. 33,

and the piles *b* are driven down behind it. A wooden mallet is used to drive the piles down, and great care must be taken to keep them vertical.

The piles are about 15 feet in length, 6 to 8 inches wide, and 3 inches in thickness. They are sharpened at the bottom, and bevelled slightly at the edges, so that when the circle is completed there is a close joint between them. After the whole ring of piles has been driven about 5 feet down, 3 or 4 feet of the sand is dug out from inside them; another curb, *c*, is then laid in position at the bottom of the excavation, the piles are again driven down, more sand is dug out, a fresh curb laid, and so on. When the first ring of piles has been driven down to its full length in this manner, and the sand dug out to a depth of about 12 feet, another set must be started, as the first will have reached its limit. This is done by laying a fresh curb, *e*, in the bottom of the shaft; this is made 18 inches less in diameter than the former curbs, so that, when placed inside the last curb laid, there will be a space of 3 inches between them. The second set of piles is driven down in this space, step by step, and supported by curbs in the manner indicated above until it reaches its limit, when another set will be started in the same manner, and so on, until the "stone head" or solid ground is reached.

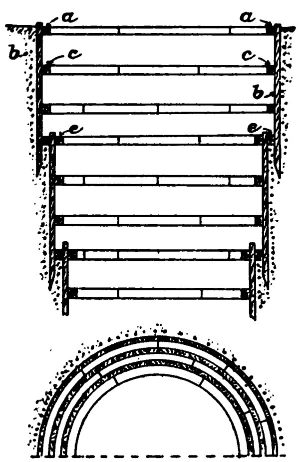


FIG. 33.—Sinking through loose ground by piles.

It will be observed that the diameter of the shaft is reduced by twice the thickness of the piles and curb for each set of piles that have to be driven, and with piles 15 feet long a fresh set is required for about every 12 feet sunk through; so that, in order to obtain the initial diameter of the excavation, the

required diameter at the "stone head" must be increased by 18 inches for every 12 feet that has to be sunk through by piling.

Piles are sometimes driven with a slight outward inclination to avoid the large reduction in size of excavation which takes place when they are driven vertically.

These inclined piles are made much smaller than the vertical ones, being only 3 or 4 feet long.

The drawback to inclined piles is that there is a small space between them, at the bottom of the length, and if the sand is very loose it leaks through into the shaft.

Sinking through Loose Ground by Drums.—Pile-driving is both costly and uncertain when the beds of loose ground are of great thickness, in which case the sinking is better performed by forcing a drum or cylinder of iron, timber or brickwork right through the bed of sand into the hard rock beneath it.

The most suitable cylinders are made of cast or wrought iron, as when timber and brickwork are used, the cylinder walls have to be very thick to resist the great pressure to which they are subjected. The operation of sinking a cast-iron drum is conducted as follows.

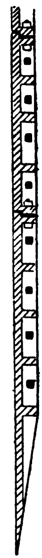


FIG. 34.—Sand tubbing.

The cylinder is built up of cast-iron plates, heavily ribbed and bracketed; these plates are smooth outside, to enable them to slip easily through the sand, and are bolted together inside, as shown in Fig. 34. The bottom of the lowest ring of plates is brought to an edge to form a cutter, as shown in the figure. The joints between the segments are made watertight by strips of soft lead-sheeting, the edges of the plates being planed. To sink the cylinder, the surface is first levelled, and the bottom ring built up by bolting the segments together. The sand is then dug out from the

inside of the ring, which gradually sinks, its weight forcing the cutter through the soft sand. After it has sunk a foot or two, the second ring is bolted on to its top, more sand is dug out to ease the passage of the tubbing. Again a fresh segment is bolted on at the top, and so on until the hard ground is reached. Should the cylinder stick, as it usually does when the sand is of great thickness, it may be either forced down from the top by hydraulic jacks, or it may be weighted with pig-iron, placed on girders put across the cylinder, and supported by the flanges. The most difficult part of the operation consists in keeping the cylinder vertical; it does not usually sink gradually, but in a series of jerks, and unless great care is taken, one side may sink faster than the other. To guard against this, the cylinders may be either hung from strong timbers by four lifting screws, or a timber guide frame may be built up on the surface. In some cases the sand is dug from inside the cylinder by hand, the water being pumped out to allow the men to work. But in case the sand is very quick, Priestman's or other excavators are employed, and then the water can be left in. These excavators or dredgers are hung on ropes or chains and dropped heavily into the shaft; as they are pulled up they close, when they are raised by a small engine, bringing the sand with them. Care must be taken not to allow the excavation to get in advance of the cylinder, for if this happens, the sand and water are liable to burst out from the sides and bottom and throw the cylinder out of plumb.

The general arrangement of sinking a cast-iron drum or cylinder will be understood from Fig. 35. *a* is the guide frame, which will be about 6 or 8 feet high; *b* is one of the hydraulic jacks; and *c*, the dredger. When the hard ground is reached, the cutting edge may be removed, and the permanent lining of the shaft brought up to the cylinder, or iron plates may be placed under the cutter for it to rest on, and the permanent lining carried right up the shaft inside the cylinder. The great advantage of cast-iron cylinders is that they occupy very little space; so that if it should be necessary to sink a second cylinder inside the first, as sometimes happens, it can

be done without greatly reducing the diameter of the shaft. Cast iron, however, has one rather serious drawback—that is, its liability to crack. Wrought iron, steel, or brick cylinders may be bent or squeezed out of shape with the pressure, but they seldom actually break.

Wrought-iron or Steel Cylinders.—These are composed of iron or steel plates about half an inch in thickness, riveted together, with an inside lining of thick brickwork. The brick-

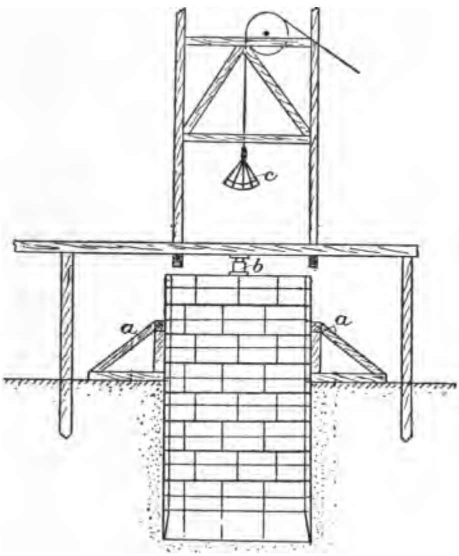


FIG. 35.—Sand tubbing—general arrangements.

work is built up on a ledge inside the cylinder and a few feet from the bottom. The steel plates are doubled at the cutting edge, and the inner plate is belled inwards till it reaches the inside of the ledge upon which the brickwork rests. This prevents the brickwork offering a flat surface to resist the sand, and so aids its passage downwards. The process of sinking these cylinders is similar to that described above.

Brick and Timber Cylinders.—These are constructed in the

following manner: A timber curb about 18 inches wide is laid down upon the site of the shaft, and brickwork is built upon it. When the brickwork has reached a height of 3 or 4 feet, a second curb is laid on the top of the brickwork, another 3 or 4 feet of brickwork is built up, a third curb is laid on its top, and so on.

The whole fabric is bound together by iron tie-bolts passing from curb to curb, and the back of the brickwork is closely lined with planks, in order to form a watertight casing.

To form a cutting edge, the lower curb is bevelled off, and a steel plate is nailed behind it. Brick cylinders are sunk in the same manner as iron ones, the brickwork and curbs being added at the top as the cylinder sinks.

Poetsch's Method of freezing Shafts.—This method of sinking through thick beds of very wet quicksand has met with great success; it is one of the recognized systems of sinking on the Continent, and has recently been employed in some very difficult sinkings in the North of England.

The principle of the Poetsch system is to freeze the ground around the shaft into a solid block, the effect of this being to consolidate the sand and hold the water back whilst the shaft is being sunk and lined through the wet ground.

The first step in sinking by this method is to put boreholes down in a circle all round the shaft, and in some cases inside it.

The number of boreholes varies according to the size of the shaft and nature of the ground; about 20 are usually required, and they are put down about a couple of feet outside the circumference of the shaft. After the boreholes are completed, two tubes are inserted in each, one tube being placed inside the other; the bottom end of the outer tube is closed, but the inner tube is left open. To freeze the sand, brine at an extremely low temperature, from −5 to 5 degrees Fahr., is circulated through the boreholes, being pumped down the inner tubes, and back to the surface up the outer ones; small cylinders of frozen ground are soon formed around each

borehole, and these gradually increase until they join, and finally a solid wall of frozen ground encircles the whole shaft. To reduce the temperature of the brine, ammonia is compressed and passed through a condenser, where it is cooled by water and is in the form of a liquid. From the condenser it passes into the refrigerator, where it expands and changes again to a gas, the expansion resulting in an extremely low temperature.

The brine is kept circulating through the refrigerator and boreholes by means of a small pump, the heat it gains in passing through the boreholes being absorbed as it passes through the coils in the refrigerator.

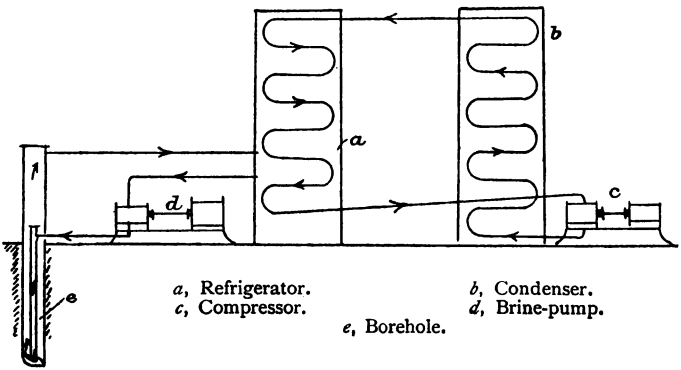


FIG. 36.—Poetsch method of sinking shafts.

Fig. 36 shows diagrammatically the circulation of the ammonia and the brine.

The following description of the sinking of the Washington shafts in the county of Durham by this system is abstracted from a paper by Mr. Ford (*Trans. I.M.E.*, vol. xxiv.):—

The strata sunk through consisted of from 80 to 90 feet of wet sand and boulder clay.

Two shafts were sunk, the winding shaft having a finished diameter of 14 feet. To get the position of the boreholes, a circle was drawn with a 10½-feet radius, and the boreholes, twenty-two in number, were set off at equal distances around its circumference. No holes were bored inside the shaft itself. To bore

the holes, tubes 6 inches in diameter were forced through the sand with screwjacks. When they reached the clay, the holes were continued with chisels and rods in the usual manner. Much difficulty was experienced in keeping the holes straight in passing through the boulder clay, as they were apt to be deflected when they struck a large boulder.

The vertical direction of the holes was tested from time to time by lowering plumb-lines down them. As soon as the holes reached their intended depth, the freezing-tubes were inserted, and the 6-inch tubes withdrawn. Special precautions were taken to make the joints of the outer tube perfectly tight, so that no leakage could take place through them.

The refrigerating plant for the two shafts consisted of the following :—

Two horizontal compressors, compressing the ammonia to 150 lbs. per square inch.

Two condensers, each 10 feet high by 5½ feet in diameter, containing 1600 feet of 1-inch tubes for the ammonia, around which were circulated 4000 gallons of water per minute.

Three refrigerators, each 10 feet high by 7 feet in diameter, containing 2000 feet of 1-inch tubes, around which the brine was circulated.

One brine pump, with two 6-inch rams, producing a flow of 144 gallons of brine per minute through the refrigerators and boreholes.

The refrigerating agent was anhydrous ammonia, and the brine consisted of a solution of chloride of magnesia.

The excavation of the frozen ground was begun forty-three days after the freezing commenced. The sand was easily got through, but the boulder clay proved more difficult, as blasting had to be resorted to, and especial precautions had to be taken so as not to disturb the freezing-tubes.

As there were no boreholes in the shaft itself, a soft core of unfrozen ground remained in its centre. The permanent lining consisted of brickwork set in cement. The brickwork consisted of two 9-inch rings, having a space of 2 inches between them, which was filled in with cement. To prevent the

cement mortar from freezing, the mixing water contained 7 per cent. of caustic soda.

The special difficulties in connection with the Poetsch system are as follows:—

(1) When the boreholes have to be very deep, it is very difficult to keep them straight, and if they get much out of plumb, wide spaces may be left between them which cannot be frozen.

(2) If the outer tube leaks, the brine escapes into the sand and prevents it from becoming frozen.

(3) Where quicksand is met with at a considerable depth from the surface, the whole of the ground right down to it would have to be frozen.

Gobert's System of freezing Shafts.—This is an important modification of Poetsch's system, the essential difference being that no brine is employed.

Boreholes are put down, as in Poetsch's arrangement, and are fitted with two tubes. The outer tube is similar to Poetsch's, but the inner is serpentine in form, and is provided with apertures at intervals in the whole of its length. The ammonia is compressed to a liquid, and forced down the inner tube; as it escapes through the apertures it vaporizes, owing to the heat of the strata, and to the pressure upon it being reduced. It is then drawn back by the gas pump, compressed again to a liquid, and so on. In the act of vaporizing—which takes place in the borehole—intense cold is produced, and this freezes the strata around the boreholes. The advantages claimed for this system are—The danger from brine leaking into the strata is avoided; and, where necessary, the lower part of a shaft can be frozen without freezing the upper portion.

Sinking by the Aid of Compressed Air.—In this method of sinking, a cast-iron cylinder is sunk through the beds of sand in the usual way. The cylinder is closed at the top and compressed air is forced in, to keep back the water,

and to enable the men to work in the pit bottom. The essential features of the system are illustrated in Fig. 37, which shows a section through a shaft in course of sinking by this process.

a is a timber frame erected upon the surface, and forming a guide to keep the cylinder vertical. *b* is the outer or main cylinder; it is built up of cast-iron segments bolted together in the usual way. *c* is an inner tube, kept always under pressure by compressed air; it is belled out at the bottom as shown, to enable the men in the shaft to work right under the main cylinder walls. *d* is an air-lock, having double doors, through which men and material pass up and down the shaft. It is made of small diameter, so as to reduce the loss of air as much as possible. The cylinder is forced down by the weight of iron or bricks placed between *b* and *c*, and, if necessary, by hydraulic jacks applied at the top. The men who work in the bottom assist the descent of the cylinder by removing any boulders or hard material which may be found under the cutting edge.

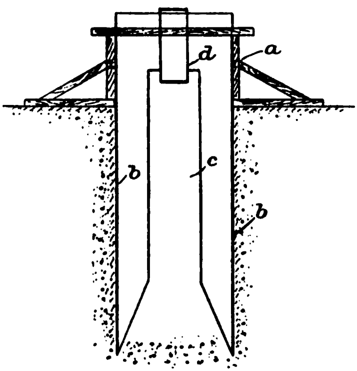


FIG. 37.—Sinking through loose ground by the aid of compressed air.

The water is kept from coming into the cylinder solely by the pressure of the air, hence it follows that the air must be at a greater pressure than that of the water.

The pressure of the water depends upon its vertical head, or depth. One cubic inch of water weighs 0·03617 lb., so that a column of water 1 inch square and 1 foot long weighs $12 \times 0{\cdot}03617$ lb. $= 0{\cdot}434$ lb. This gives us the rule that the pressure in pounds per square foot due to a head of water = head in feet $\times\ 0{\cdot}434$.

It has been found that the greatest pressure under which men can work is about 45 lbs. per square inch, and this corresponds to a head of $\frac{45}{0 \cdot 434} = 103 \cdot 6$ feet, so that the greatest depth to which a shaft can be sunk by this system is about 100 feet.

Tunnels under river-beds are sometimes driven by this method, but it has not been largely applied to the sinking of shafts.

One of the drawbacks of this method is that when the pressure of the air is high, the men employed in the cylinder can only work very short shifts; in some cases they have had to be changed every two hours.

CHAPTER VI.

SINKING (*continued*).

Surface Arrangements.—After the "stone head" has been reached by one of the methods described in the last chapter the lining of the shaft is carried up some distance above the surface to provide a tipping ground for the sinking-dirt. If the ground is strong right up to the surface, a few yards are usually sunk and bricked before the engines and pit top are erected; solid brickwork should be built up around the shaft

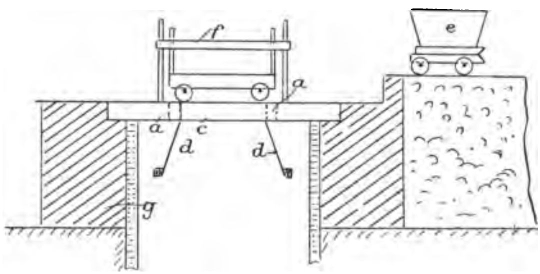


FIG. 38.—Arrangements at a sinking pit top.

as shown at *g*, Fig. 38. This forms the foundation for the headgear, and archways must be provided where necessary for fan drifts, haulage ropes, steam and water pipes, etc.

The pit top is usually arranged as in Fig. 38. A frame, consisting of four strong timbers, *aa*, is placed across the shaft, and the whole boarded over except the square aperture *c*. Guide-planks, *d*, are set at an angle, as shown in sketch; and

a lorry, or rolling bridge, arranged to run on rails, so that it can be made to either cover the shaft or leave it open. In the figure the shaft is shown covered, but the lorry can be run back clear of it. A fence, *f*, is erected to guard the mouth of the shaft; its sides and back are fixed, but the front is attached to the lorry, and moves with it.

Lifting doors are sometimes used instead of the sliding lorry; they are arranged as in Fig. 39, and, when open, form the fence to two sides of the shaft, as shown by the dotted lines. These doors can be raised by a hand lever when they are properly balanced, or they may be operated by a steam cylinder.

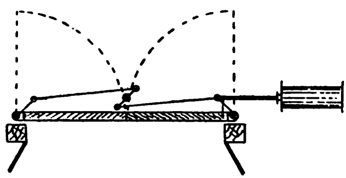


FIG. 39.—Lifting doors for sinking pit top.

Winding Engines for Sinking.—These are usually a pair of horizontal engines, having cylinders from 16 to 20 inches in diameter. To enable them to lift heavy loads the drum should be on the second motion, the ratio of gearing being about 2 or 3 to 1. Engines for sinking must be easily handled, run steadily, and be provided with a powerful foot brake. If the shaft is to be very deep, the main winding engines are frequently erected whilst the earlier part of the sinking is in progress, and utilized for the latter part.

Capstan Engines.—In all important sinkings—especially where large volumes of water have to be dealt with—very heavy weights have to be raised and lowered in the shafts, and for this purpose a capstan is required. A capstan engine should have several strong cast-iron drums, any of which can be worked whilst the others remain stationary. The gearing may be about 25 or 30 to 1; that is to say, the engines make 25 or 30 revolutions to each revolution of the drum, so that, whilst the engines are travelling at a high speed, the drums are moving quite slowly, the reduction in speed being compensated by a corresponding increase in lifting capacity. The following example shows the manner of calculating the weight

that a pair of capstan engines should lift: A pair of capstan engines having cylinders 12 inches in diameter by 24 inches stroke are geared down to drums 4 feet in diameter, the ratio of the gearing being 25 to 1. Calculate the weight they should lift when the average steam pressure in the cylinders is 40 lbs. per square inch; and find also the speed at which such weight would be raised, when the engines are making 75 revolutions per minute.

First, find the area of the cylinders. This is done by squaring the diameter and multiplying by 0·7854; so that the area of each cylinder is $12 \times 12 \times 0{\cdot}7854 = 113{\cdot}1$ sq. inches, and as there are two cylinders, their combined area is 226·2 sq. inches. On each square inch there is a pressure of 40 lbs., hence the total pressure on the pistons is $226{\cdot}2 \times 40 = 9048$ lbs.

The engines make 75 revolutions per minute, and as their stroke is 2 feet, each piston travels 4 feet per revolution; so the piston speed is $75 \times 4 = 300$ feet per minute. The ratio of gearing is 25 to 1; therefore the number of revolutions that the drum makes per minute is $\frac{75}{25} = 3$.

The diameter of the drum is 4 feet, and the circumference of a circle is found by multiplying its diameter by 3·1416; this makes the circumference of the drum $4 \times 3{\cdot}1416 = 12{\cdot}56$ feet. As the speed of the rope is exactly the same as the speed of the circumference of the drum, the speed at which the rope, and the weight upon it, travels is $3 \times 12{\cdot}56 = 37{\cdot}68$ feet per minute. Now the pressure on the piston is 9040 lbs., and the space it moves through per minute is 300 feet, whilst the weight lifted only travels through 37·68 feet; so that the weight the capstan should raise bears the same proportion to the pressure on the piston as the distance the weight moves bears to the distance the piston moves; or as 37·68 : 300 : : 9048 to weight lifted; or, in other words—

$$\frac{\text{pressure on piston} \times \text{distance it moves}}{\text{distance weight moves}} = \left\{ \begin{array}{l} \text{weight capable of} \\ \text{being raised} \end{array} \right.$$

so that the weight raised should be $\frac{9048 \times 300}{37{\cdot}68} = 72{,}038$

A large deduction must be made from this, because a considerable amount of power is required to overcome the friction of the engines and gearing. This deduction may be taken at one-third of the whole; so that the engines would raise about 48,000 lbs. or over 21 tons at a speed of 37·68 feet per minute.

Ropes used for Sinking.—The special requirements in ropes used for sinking are that they must not "spin" or twist. Ordinary steel ropes are usually employed, but locked coil ropes are preferred by many, because they run more steadily than ropes made of round wires. Flat ropes have been used, but they have not met with any great success, for if they do spin at all their oscillations are very violent.

Calculations as to the weight and breaking strain of wire ropes are given in Chapter XXIII.

Ventilation.—Sinking pits are ventilated by small fans. A line of air-pipes made of sheet-iron, and 18 or 24 inches in diameter, is spiked on to the side of the shaft, down which the air is forced, or up which it is drawn by the fan, the former by preference, as the bottom of the shaft is more efficiently ventilated. Ample ventilating power should be provided, as it is necessary to clear the shafts very quickly of the smoke given off when a heavy round of shots is fired; moreover, gas is sometimes met with in considerable quantities, especially in the vicinity of coal-seams.

Lighting.—Important sinking pits are usually lit by electric lamps. When gas is made safety lamps only should be employed. A recent disastrous explosion in a sinking pit was attributed to the flame given off by an electric-lighting cable, when it was accidentally severed by being struck with a shovel.

Winding the Débris.—The stone is wound to the surface in "hoppits" or "kibbles;" Fig. 40 shows their usual form. The dimensions given are about the average, but the tendency is towards increased size. The bow of the hoppit is pivoted on trunnions, *a* in sketch. These trunnions are set below the centre of gravity, so that the hoppit can be easily tipped right over by withdrawing the cotter *b*, and lifting the hoop *c* clear

G

of the pin. The hoppit is attached to the rope by means of a spring hook and swivel.

To wind the *débris* two hoppits are required. One is always in the shaft bottom being filled, whilst the other is travelling the shaft and being emptied. The full hoppit is drawn a few feet above the surface, the rolling bridge carrying a tip waggon is run over the shaft, and the contents of the hoppit emptied into it. The rolling bridge is then run back clear of the shaft, and the hoppit lowered to within a few yards of the bottom, and held there by the engine-man until it is signalled

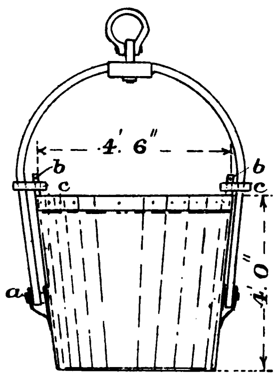


FIG. 40.—Hoppit for sinking.

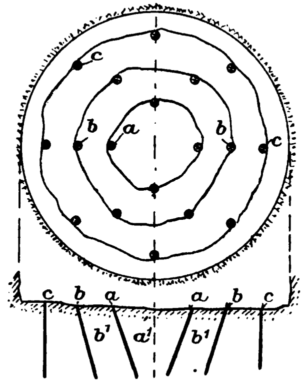


FIG. 41.—Arrangement of shots in sinking.

for by the sinkers, two of whom await its arrival, and push it into a convenient spot. The winding rope is then uncoupled and attached to the full hoppit, which is raised about 6 feet; it is there steadied by the sinkers, who signal it away, after satisfying themselves that it is properly loaded and has no loose stones adhering to the sides or bottom.

Excavation.—The whole of the excavation is effected by blasting. The shot-holes are usually bored by striking-drills. The drills are made from octagonal steel, forged chisel-shape at the ends—three of varying lengths forming a set. One man

holds the drill, and one or, in hard ground, two men strike the head with heavy hammers. At every blow the drill is turned through a small angle, great care being taken to keep the holes round and of even diameter all through. The holes are cleaned out by scrapers, and, if dry, water must be poured down to keep the drills cool, and bind the dust so that it can be cleaned out by the scraper.

The shot-holes should be arranged as in Fig. 41. The ring of holes marked *a* is first fired. These holes are termed "sumpers," and are deeper and more heavily charged than the others; they should blow out that portion of the ground marked *a'*. After they are fired the loosened material is filled out, and the second ring, *b*, is drilled, charged, and fired. This ring of shots should lift the ground marked *b'*, and make a "loose end" for the next ring, *c*.

The best explosive for hard ground is dynamite or blasting gelatine; owing to their high specific gravity and great strength, much smaller holes are required for these explosives than would be necessary for blasting powder. Moreover, the work required of an explosive is not merely to lift the rock, it should also break it up into pieces small enough to be conveniently handled by the sinkers.

Shots should be fired by an electric battery, through a cable taken from the surface. The battery should be strong enough to fire several shots simultaneously, to reduce the time that is lost by withdrawing the men from the shaft during the process of shot-firing. When blasting powder is employed or the shots are fired by ordinary fuses, the fuses of shots which are fired together should always be cut of different lengths, so that the number of detonations can be counted; otherwise, a shot might hang fire without being detected, and explode whilst the sinkers were at work.

Method of temporarily supporting the Shaft Sides. —Formerly it was the custom to leave the hard beds of rock unsupported until the permanent lining was built, but it is now becoming usual to support the whole of the sides of the

excavation as the sinking proceeds, because the hardest of rocks may be shaken by the blasting, and portions may break away without warning.

Under ordinary conditions the shaft sides are temporarily supported by iron rings and planks, all of which are drawn out as the permanent lining of brickwork or tubbing is built up. The arrangement will be understood by reference to Fig. 42, which shows two views of part of a sinking pit timbered with planks and iron rings. The rings marked *a* are made of flat

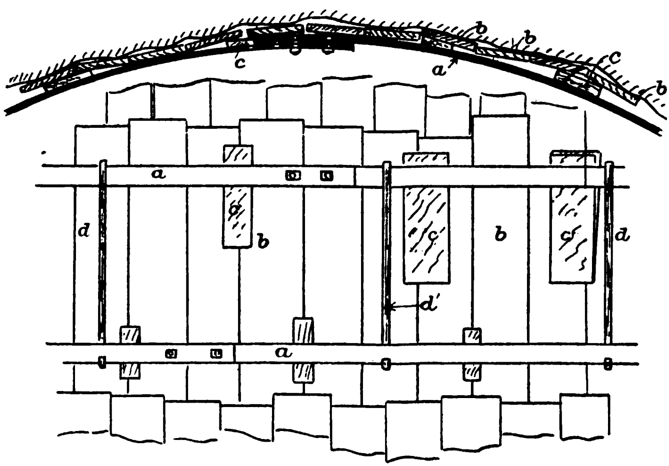


FIG. 42.—Supporting the sides of a sinking pit.

iron bars from 3 to 4 inches wide by $\frac{3}{4}$ inch thick; they are made up of segments, which have several bolt-holes in each end, so that the size of the rings can be varied to suit the irregularities in the uneven shaft sides. The planks *b* are 6 or 7 inches wide by $1\frac{1}{4}$ inch thick; they are set against the strata all round the shaft, and held in position by being wedged between the sides and the rings, the wedges *c* being used for that purpose. The rings are kept in place by the hangers *d*, made of iron about 1 inch square, their distance apart varying according to the nature of the ground. Formerly wooden

rings, termed curbs, made of oak about 6 inches square, were employed instead of iron rings. They are stronger than the iron ones, and more costly, but not so convenient, and are now not often employed, except where the ground is unusually heavy.

Bricking the Shaft.—Shafts are usually lined with bricks of the ordinary size, that is 9 by 4 by 3 inches, moulded to the circle of the shaft. In some cases large fireclay lumps are employed, but well-burned red bricks give satisfactory results except under exceptional circumstances.

The brick lining is built up in sections, the operation being conducted as follows: The shaft—which has, of course, been sunk large enough to allow for the brickwork—is reduced to its finished size, leaving a ledge all the way round it; this ledge, which is termed the *curb-bed*, forms the foundation upon which the brickwork is built. It is most important that the curb-beds should be perfectly level, and that their centres should be the exact centre of the shaft; to ensure this they must be carefully tested by means of the spirit-level and centre line.

The *curbs* may be made of oak or of cast iron, the latter being preferred for wet shafts, as oak is liable to rot. Cast-iron curbs vary from 12 to 16 inches in width, and are about $1\frac{1}{2}$ inch thick, made in eight or ten segments.

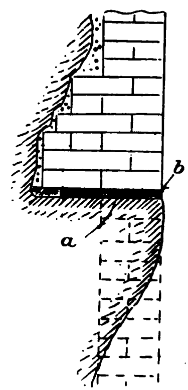


FIG. 43. — Section through brickwork and curb.

The curb is laid on the bed, and wedged tightly from behind with hard wood wedges; whilst being wedged up, it should be tested with spirit-level and centre line, great care being taken to lay it perfectly true. Fig. 43 is a section through a curb-bed and short length of brickwork; *a* is the curb-bed, and *b* the curb. The shaft is shown to be belled out a little at the curb-bed; this is necessary, because the curb is made wider

than the thickness of the brickwork. In the sketch the brickwork is shown to be of more than ordinary thickness at the curb. This is not always the case, but it is good practice, as, if the shaft should at any time run in, these thick belts of brickwork would probably limit the injury to one length. When sinking is resumed below the curb-bed, the shaft is gradually belled out to its full diameter, leaving a bracket of rock all round it to support the curb.

As a length of brickwork approaches the curb above, the bracket of rock is removed in short lengths, and the brickwork carried right up to the underside of the curb, or the curb may be left partly supported on natural ground.

The centre line is a galvanized wire cord carrying a heavy weight. When it is required, a beam is placed across the shaft at the surface, from a mark on which the centre line is hung. There is no necessity to use the centre line except when setting a curb, as the brickwork is kept vertical by lines hung from curb to curb. Foulstone's centre-line apparatus consists of a girder which can be run out over the shaft by means of a rack and wheel. The centre line is hung over a pulley at the end of this girder, and can be raised or lowered by means of a small winch. When not in use the centre line is hung down the shaft, but racked back close to the side, so that when it is required it has only to be racked forward to the centre and, if necessary, lowered a few feet. This arrangement saves considerable time, as in a deep shaft it may take an hour or more to lower an ordinary line, and to steady it after it is lowered.

A good curb-bed is not always obtainable. If the ground is too soft to afford the necessary support, the curb may be hung on chains from baulks fixed in the shaft some little distance up. A better method is to support the curb by driving in iron plugs all round the shaft side, and laying the curb upon them. Sometimes a square or octagonal timber frame is let into the sides, and the curb built upon it.

Scaffolds.—As the brickwork is being built, the workmen

require a scaffold to work upon. Fig. 44 shows the ordinary half-moon scaffold. It is made in two pieces for convenience in lifting, and is constructed of 3-inch planks bolted to a strong framework. It is provided with a flap to enable it to pass the air and water pipes, the whole forming a circle 6 or 8 inches less in diameter than the shaft.

The scaffold shown in the figure is supported by the bolts *b*. These are driven out and rest on the top of the brickwork, which is then built up for another 4 or 5 feet. The scaffold then has to be raised. To do this, the rope from the capstan or winding engine is hung on to the bridle chains attached to the scaffold, the bolts are knocked back, the scaffold raised just above the brickwork, and supported thereon by the bolts, which are again driven out.

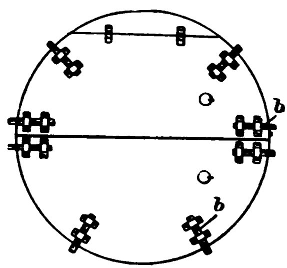


FIG. 44.—Sinking scaffold.

The scaffold is frequently carried by chains instead of upon bolts, these chains being hung from timbers placed across the shaft. One very neat arrangement is to hang the chains from the curb above. When this is done, the curbs are built upon four pieces of flat bar iron, which are bent back behind the curbs. These bars are spaced at equal distances around the shaft, and their ends, which project a few inches from the curb, are forged into bows, from which chains are suspended. The chains hang down the shaft close to the sides, and are provided at equal distances with large links, from which the scaffold chains are hung.

When a half-moon scaffold is not in use, it may either be hung up in the shaft in halves, or it may be taken to bank. Some consider the latter the safer, as if anything should fall down the shaft and hit a scaffold suspended in it, it might throw the scaffold down the pit. This happened some years ago at a Derbyshire colliery.

A very good scaffold is shown in Fig. 45. It consists of

a strong circular frame made of timber, having a hole 6 or 8 feet in diameter in the centre. This central aperture can be closed, when desired, by a timber platform. It is hung from two ropes wound round the drums of a capstan engine. These drums are capable of independent motion, to enable the scaffold to be kept straight. The outer portion, *a* in the figure, is always left in the shaft, and when sinking is in progress it partially shelters the men from anything that might fall down.

When sinking is stopped, and the brickwork or tubbing is

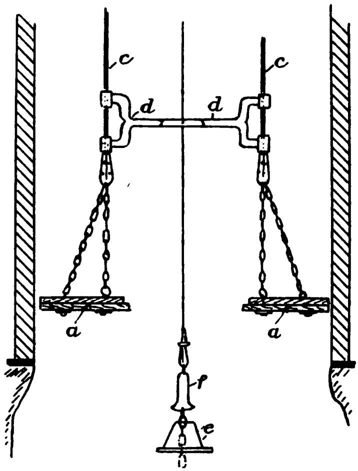


FIG. 45.—Sinking scaffold.

being put in, a central piece is dropped into position, forming a solid platform upon which the men work. The whole scaffold can be very easily raised or lowered by means of the capstan, without withdrawing the men. The two capstan ropes *c*, in addition to carrying the scaffold, form guides to steady the hoppit when in the shaft, the arrangements for guiding the hoppit being as follows: *d* is a wrought-iron guide bar, having a pair of slides or thimbles at each end, and an aperture in its centre just large enough to pass the detaching hook. At the end of the winding rope is the cone *e*, which is rather larger

in diameter than the widest part of the detaching hook *f*. When the hoppit is in the shaft, the guide bar *d* is carried by the shoulders of the cone *e*. As the hoppit descends it leaves the guide bar at the scaffold, and picks it up again on its return journey.

As the brickwork is built up and the scaffold gradually raised from the bottom, the space below is sometimes allowed to fill with water. When this is the case, care must be taken that the water does not cut off the ventilation; if it rises above the bottom of the air-pipes, a joint must be broken above the water-level.

Sinking and Bricking simultaneously.—When sinking and bricking are carried on as described above, the sinking must be stopped whenever a length has to be bricked; this is the usual practice, but several shafts have been sunk and bricked simultaneously, thereby avoiding the delay occasioned by the stoppage of the sinking whilst the lining is being built. Galloway's scaffold for sinking and bricking simultaneously consists of a platform having a circular aperture, through which the hoppit to serve the sinkers passes. This opening is surrounded by a circular iron fence 7 or 8 feet high, which prevents men and materials falling from the bricking platform on to the sinkers. The bricklayers work on the annular space between the fence and the shaft-sides; they are served by a separate engine, and protected by a sheet-iron roof. The scaffold is suspended in the shaft, and raised or lowered by capstan ropes, which are arranged to form guides for the hoppits serving both sinkers and masons.

Sinking with Rock Drills.—These are usually driven by compressed air, and may be either rotary or percussive. For sinking purposes, percussive drills driven by compressed air are almost exclusively employed. A rock drill of this description consists of a cylinder 3 or 4 inches in diameter, with a 5 or 6 inches stroke. In this cylinder is fitted a piston having a very strong rod, into the end of which is fixed the drill. Compressed air is admitted into each end of the

cylinder alternately, giving the piston and the drill which it carries a rapid reciprocating motion. At each stroke the drill is turned through a small angle by means of a rachet arrangement fitted into the top of the piston. As the drill works and the hole becomes deeper, it is fed forward either automatically or by hand, the latter by preference, as there are fewer complications, and the rate of feed can be varied to suit the different kinds of ground which are met with. Rock drills are either mounted on tripods, or upon a frame fixed in the shaft. There are many types of rock drills, differing chiefly in their valve gear, which is usually actuated by some form of tappit struck by the drill at either end of its stroke.

For very hard rock the drills should be + or X shaped. Those shaped thus + are the more easily sharpened, but the X drills are better suited for extremely hard ground.

In sinking with rock drills, all the holes in a round should be drilled in a definite pattern, which should be so arranged as to clear out a given length of ground. The inner ring, known as "sumpers," should be first simultaneously fired by an electric battery; the dirt is then filled out, the second ring fired simultaneously, and so on.

Fig. 46 shows Walker's patent drill frame arranged for rotary drills. *a* is a stand of cast iron, forming a receiver for compressed air, which is supplied to it through a pipe from an air compressor on the surface. At intervals along the outside of the stand are eight rotary motors, *b*, *b*, which are driven by compressed air, taken from the receivers through the pipes *c*.

The radial arms *d*, *d* are fixed to the top of the stand, and are provided with a long slot, in which the drills *e*, *e* can be secured in any position. The arms are adjustable, to enable them to be run out against the sides of the shaft in order to hold them firmly in position. The drills are driven by the motors through the flexible shafts *f*, *f*, or, if desired, any of them can be worked by hand through rachets. To raise or lower the frame in the shaft, a capstan rope is attached to the chains, and the arms *d*, *d* are swung round into a vertical position. The lower part of the receiver *a* contains water,

which, being under pressure, can be used to flush the drill-holes.

Walker's patent drill frame for use in conjunction with percussive drills consists of a heavy cast-iron ring-shaped frame carrying adjustable arms, upon which the drills can be

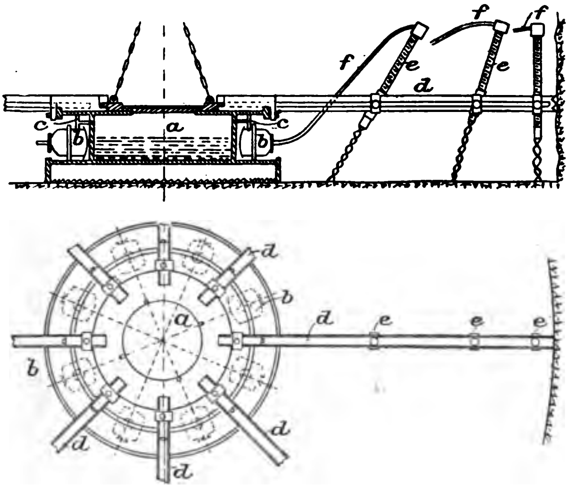


FIG. 46.—Walker's patent sinking frame.

attached in any desired position, the compressed air being taken from a circular receiver fitted to the underside of the frame.

Rock drills are of no great advantage except in very hard ground, where they affect great economies, as the holes can be made deeper and much more quickly than by hand.

CHAPTER VII.

SINKING (*continued*).

Presence of Water in Shafts.—Some of the beds forming the earth's crust are porous, and some impervious. The porous rocks may hold large volumes of water, especially when overlain and underlain by impervious or watertight beds of clay or

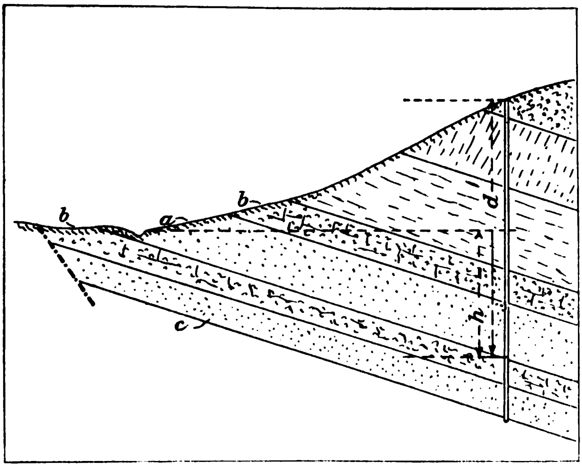


FIG. 47.—Occurrence of water in sinking pits.

shale. The manner in which water makes its way into sinking pits is indicated by Fig. 47. *a* is a thick porous rock outcropping in a valley, both overlain and underlain by the impervious beds *b*, *b*. A stream is shown to traverse the outcrop

of the porous rock, and water from it will percolate into the rock and saturate the whole of its pores. If no stream be present, the rainfall may be quite sufficient to form a heavy feeder of water. As soon as the rock *a* is struck in the sinking shaft, a feeder of water will be encountered, which, if not pumped or tubbed back, will rise in the shaft to the level of the outcrop of the bed.

Another bed of rock is shown at *c*, but this does not outcrop, as it is thrown out by a fault. It is reasonable to expect that no permanent feeder of water will be met with in passing through *c*, as there is no source of supply; but the rock may contain a store of water, which will gradually diminish if pumped. Faults usually form a barrier against water, especially when they throw a porous rock against an impervious one.

It will be noticed, from the above, that water may be met with either as a permanent feeder or as a pound or reservoir. The former should be tubbed off if possible, but the latter may be pumped, as it will gradually diminish, and finally cease altogether.

Tubbing.—The operation of tubbing consists of inserting a watertight lining through the water-bearing strata in such a manner as to dam back any feeders of water which may be present.

Tubbing usually consists of cast-iron segments, but when the pressure of the water is not very great, brickwork and cement may be used. A segment of cast-iron tubbing is shown in Fig. 48. The segments are usually about 4 feet in length, 2 or 3 feet in depth, and of a thickness varying according to the pressure to which they are to be subjected. The flanges are made from 4 to 5 inches wide, the top and one side flange being provided with a projecting rib to keep the adjoining segments in position and to provide something to wedge against. Each segment is divided into panels by cross-ribs, and one or more holes are cast in each to enable the water to escape from behind the tubbing whilst it is being

put in and wedged. The thickness of metal tubbing depends upon the diameter of the shaft and upon the pressure of the water which it has to keep back, and as this latter depends upon the vertical head, it follows that the thickness of the tubbing should also vary with the head of water.

As it is not always possible to ascertain the exact head of a feeder, it is usual to take the head as being equal to the depth of the shaft. That this is not necessarily correct is shown by Fig. 47, where *h* is the head of water, and *d* the depth of the shaft. Had the shaft been sunk in a valley, and

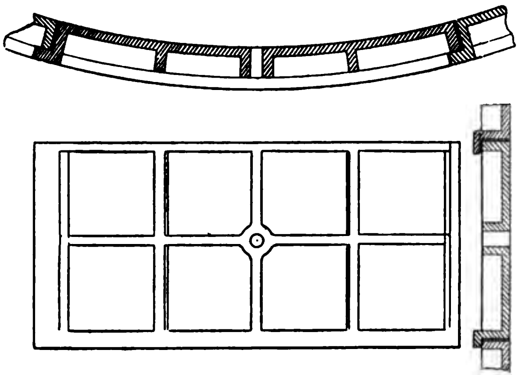


FIG. 48.—Cast-iron tubbing.

the feeder taken its rise from higher ground, the head of water would have been more than the depth of the shaft.

There are several formulæ for calculating the thickness of tubbing; the one given below is André's, which is one of the best known, and conforms fairly well with modern practice.

This formula is not scientifically correct, as it does not take the horizontal flanges and ribs into account, and they add enormously to the strength of the tubbing.

Where T = thickness of plates in inches,
H = head of water in feet,
D = diameter of shaft in feet,

$$T = 0{\cdot}35 + 0{\cdot}00025\,HD \quad (\textit{André})$$

Example.—Head of water, 250 feet; diameter of shaft, 16 feet. Find thickness of tubbing.

$$0{\cdot}35 + 0{\cdot}00025 \times 250 \times 16 = 1{\cdot}35 \textit{ inch.}$$

This is the thickness at the bottom of the tubbing. If it extended right to the surface, its thickness should diminish as the head becomes less. About four or five lengths of different thicknesses would be employed, each being $\frac{1}{8}$ inch thinner than the one below. A length of tubbing is put in as follows:—

A curb-bed is prepared some little distance below the bottom of the water-bearing rocks in carefully selected ground, and is dressed perfectly level and concentric, no blasting being allowed anywhere near it. After the curb-bed has been tested with centre line, straight-edge, and spirit-level, a thin layer of soft wood or tarred flannel is laid upon it, and on this the curb is placed in position.

The space behind the curb is packed tightly with dry wood, the grain being placed vertically. Hard wooden wedges are then driven into the packing to make an absolutely water-tight joint. The wedges are 4 or 5 inches long, 1 inch wide, and about $\frac{3}{8}$ inch thick at the upper end. They can at first be driven straight into the packing timber with a hammer, but as the wedging proceeds the packing becomes too hard to admit the wedges, and the steel chisels have to be driven in to make a way for them.

A curb is considered to be sufficiently wedged when the packing behind it is so hard that a steel chisel cannot be driven into it.

A tubbing curb suitable for a shaft 18 feet in diameter is shown in Fig. 49. It is cast hollow (the open side being towards the shaft), and strengthened by vertical ribs. The tubbing is kept in position on the curb by a ledge, as shown in figure.

Every segment of tubbing is carefully examined and tested before it is sent down the pit. The most common defects are "honeycombs," which are hollow places in the casting.

"Honeycombs" may be quite concealed from view, and to discover them the tubbing should be struck all over with pointed hammers; by doing this any defect is discovered by the sound of the blow, even if the hammer-point does not break through the crust into the hollow space.

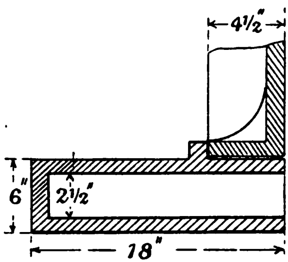


FIG. 49.—Curb for tubbing.

Each segment of tubbing is sent down the pit on a large D link, which is hung on to the winding rope, the pin of the D link passing through the centre hole in the tubbing.

As soon as the engine-man lowers the segment to within reach of the sinkers, several of them seize it and swing it into its place, the weight being carried by the winding rope. The pin of the D link is then withdrawn, and it is sent back on the winding rope for another segment. Layers of fir sheeting about $\frac{3}{8}$ inch in thickness are placed between the curb and the tubbing, also between the segments, the grain of the sheeting pointing towards the centre of the shaft. After a ring of tubbing has been placed in position, it is wedged up from behind by folding wedges, as shown in Fig. 50. These

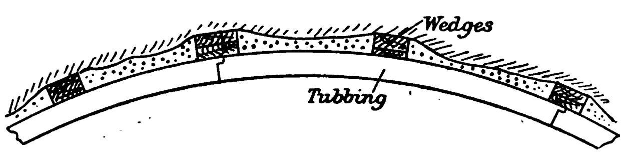


FIG. 50.—Tubbing in position.

wedges are about 2 feet long and 6 or 8 inches wide. After a ring has been wedged up, a layer of sheeting is laid on its top flange, and another ring of tubbing built up on it. The vertical joints are broken; that is, the joints between the segments in one ring come in line with the centres of the segments in the rings above and below. The tubbing is carried on in this manner, either up to the surface or to the

curb above. If it is to terminate at a curb, it is necessary to make a perfectly watertight joint between the tubbing and curb. To do this, the curb up to which the tubbing is to be carried is provided with a ledge on its lower side, similar to the ledge on the upper side of the curb shown in Fig. 49. The tubbing is carried right up to this upper curb, the top ring being specially cast of the exact height required. As it approaches the curb, sufficient ground is chipped away from underneath it to allow the tubbing to be built right up. The space between the back of the tubbing and the shaft sides is usually filled in as the tubbing is built up, concrete being employed for the first few rings, and soil or clay for the remainder.

The final operation is to wedge the tubbing. This is done by driving wooden wedges into the sheeting which forms the joints between the segments. The vertical joints are wedged first, beginning at the bottom and working upwards. Next the horizontal joints are wedged, and lastly plugs are driven into the holes in the segments and wedged up tightly, still working upwards from below.

It is usually considered necessary to take precautions to prevent air from being imprisoned behind the tubbing, as it is thought to lead to the plates being fractured or forced out of position. In some sinkings, however, no special precautions have been taken, and no damage has followed.

Coffering.—When the pressure of water is not very great, it may be kept back by brick and cement coffering. This consists of several concentric rings of brickwork, having a space between each filled in with cement. Each ring is entirely independent of the next, the vertical as well as the horizontal joints being broken.

The cement rings have to be taken up continuously, the top of the cement never being allowed to set, as, if it does, joints or faces are formed, through which the water would pass. The space between the back of the brickwork and the shaft sides is filled in tightly with sifted soil. Coffering is much cheaper than tubbing, but it is unable to resist great pressure,

and owing to its thickness, a much larger excavation is required than is necessary for cast-iron tubbing.

Tubbing is usually considered more economical than coffering when the depth of the shaft exceeds 100 feet.

Cast-iron tubbing is subject to corrosion by acid water and by the fumes given off from a furnace, when one is employed. Several cases are on record where tubbing has burst, or has become so thin as to be dangerous.

Furnaces have a very detrimental effect upon tubbing, not only on account of the wearing of the plates, but when the shaft has to be cooled down for examination or repairs, the plates contract, leading to leakages and blown-out sheeting. The tubbing of a furnace shaft is sometimes lined with brickwork. This protects the plates, but renders it impossible to examine the tubbing and wedge leaky joints.

Pumping from Sinking Shafts.—The feeders of water met with in sinking shafts have to be pumped or wound to the surface until they are tubbed off. The following are the three principal methods of doing this: (*a*) By water-barrels; (*b*) by hanging lifts worked by an engine on the surface; and (*c*) by steam or electric pumps suspended in the shaft. Each of these methods has its own advantages and drawbacks, and frequently a combination of two of them is employed.

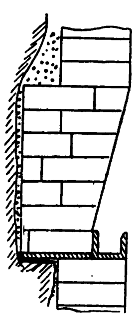


FIG. 51.—Section of water-ring.

Winding Water in Barrels.—In its simplest form this method is suitable only for dealing with small volumes of water. The water made in the shaft is caught by water-rings or garlands. These (Fig. 51) consist of a curb, the front of which is formed into a gutter or water-ring, the brickwork being set back the width of this gutter at the curb, and gradually regaining its proper position. The water which is made above the curb flows into the water-ring through holes left in the brickwork, and is led down the shaft in pipes. These pipes terminate in

a flexible hose, which is taken to the water-barrel, which is wound up when full. The water which is made in the shaft bottom is "laded" into the water-barrel by scoops or buckets.

Galloway's Pneumatic Water-barrel.—This arrangement was used at the sinking of Llanbradach Colliery, where feeders of over a hundred gallons per minute were dealt with. When water-barrels are filled by hand, only a very small quantity can be raised, but by Galloway's arrangement much larger feeders can be drawn, and at considerably less cost and inconvenience. It consists of a close-topped iron tank, into the bottom of which is fitted a large valve opening upwards. The tank is also provided with a glass gauge to indicate the depth of the water it contains, and an iron pipe, which passes through the side and rises within it almost up to the top. An air-pump is fixed upon the surface, and from it a range of 3-inch iron pipes is taken down the shaft, terminating with a flexible hose. The barrel is lowered into the water in the pit bottom, and the hose coupled on to the pipe in the barrel by means of an instantaneous coupling similar to those used for the vacuum brakes on railway trains.

The action of the air-pump forms a partial vacuum in the barrel, and water is drawn in through the bottom valve. When the tank is full, the vacuum pipes are uncoupled, and the tank is sent away to the surface and there emptied.

Hanging Lifts.—This method of pumping water from sinking pits consists of hanging bucket-pumps in the shafts, the buckets being operated through rods driven by an engine on the surface. This method is still very largely employed, but it appears to be gradually giving place to self-contained steam or electric sinking-pumps, hung in the shafts and supplied with steam or electricity through steam-pipes or cables.

The ordinary arrangement of a hanging lift is shown in Fig. 52. *a* is a set of timbers sufficiently strong to carry the whole weight of the lift, and upon *a* are mounted two sets of cast-iron pulleys, *b*, *b*. The lift *c* is carried by ground spears, *d*, *d*, which are secured to the bottom of the lift, and the whole bound together by the clamps *e*. Attached to the top of the

ground spears are two sets of pulleys, *g*, *g*, each having three sheaves, whilst the sets of pulleys *b*, *b* have each four sheaves.

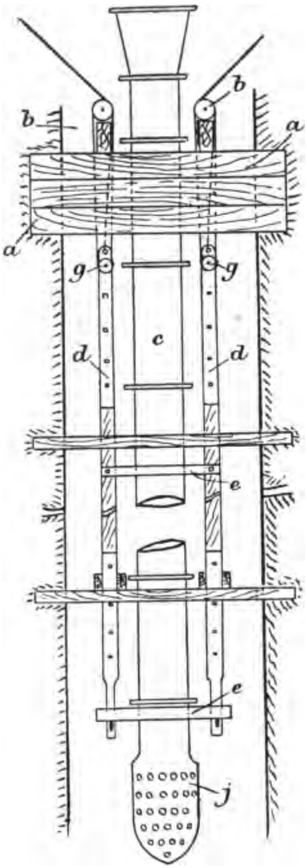


FIG. 52.—Hanging lift for sinking pit.

Block ropes are threaded through these pulleys in the usual way, and led to the drums on a capstan engine, by which the lift can be raised and lowered at will. The bottom of the lift terminates in a windbore, *j*. This, in its ordinary form, consists of a bulb cast in the bottom of the lowest pipe, having holes in it through which the water passes and is strained. A stuffing-box is also provided, by means of which the windbore can be lowered a few feet without altering the position of the pumps. As the shaft is deepened, the pumps are lowered by means of the capstan, until the blocks have been run out their full length; they are then raised, and another pipe added at the top. Additional ground spears and pump-rods are attached to the top of the others as required. The pumps are driven by bell cranks, the horizontal leg being attached to the pump-rod by a clamp, and the vertical leg to a connecting-rod, which is driven by the crank on the second-motion shaft of an engine. Sometimes the main winding engines are put down before the shaft is sunk, and utilized to drive the pumps during the

sinking. If more water is made than can be dealt with by one set of pumps, other lifts have to be added.

An ordinary bucket-pump cannot work to advantage when the height of the lift is more than 80 to 100 yards, and when this is exceeded, two lifts have to be employed, the lower one, which follows the sinking down, delivering its water into a cistern, from which the upper lift pumps to the surface.

The bottom of the shaft just below the pumps is kept a little in advance, so that the water drains into it, keeping the remainder of the shaft comparatively dry. The windbore should be raised clear of the ground when shots are being fired, to lessen the risk of injury from flying masses of rock; and even when this precaution is taken, windbores are apt to be fractured.

At the Maypole sinking near Wigan, a short length of armoured hose was introduced between the pumps and windbore. This gave the suction pipe a certain amount of elasticity, and prevented its being broken by the sudden shock of the shots.

Steam-pumps hung in the Shafts.—In this method of draining sinking pits, self-contained pumps of special design are hung down the shaft on chains or capstan ropes. The advantages of this method are as follows: more water can be dealt with, as the pumps take up less room in the shaft than hanging lifts of equal capacity; the pumps are light and handy, and so can be raised or lowered easily, and swung to one side when required; each pump works independently of the others, and can be controlled from the surface. They are also more economical in first cost, as no foundations are necessary, and the pumps and pipes can be easily disposed of when they are no longer required. There are several types of pumps specially designed for sinking, the "Denaby" and "Evans" being notable examples.

Sinking-pumps have no bed-plates, but are hung in the shafts by ropes or chains. It is very important that they should occupy little room, and be reliable in working, and not easily put out of order. The steam-cylinder is fixed above

the pump, which is worked direct from the engine piston-rod. Wrought-iron are much superior to cast-iron pipes for this class of work, being much lighter, more easily fixed, and less likely to fracture.

Fig. 53 shows the arrangement adopted for hanging the sinking-pumps at the sinking of the Cadeby Main Colliery.* *a*, *a* are old steel winding ropes, which are attached to the pump, taken up the shaft, and over pulleys at the surface, to the capstan drums. *b* are stretchers made of flat bar iron, to which the delivery, steam, and exhaust pipes are secured by the staples *c*, *c*, *c*, and the ropes by the collars *e*, *e*. These stretchers are placed about 9 feet apart, and by binding the pipes to the ropes, steady the whole arrangement.

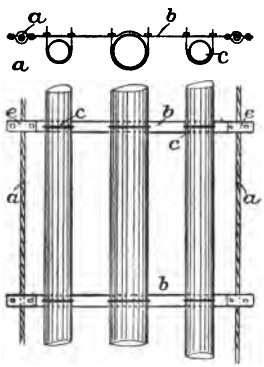


FIG. 53.—Sinking-pumps hung on ropes.

The pumps are provided with sliding suctions, and the steam-pipes are taken through stuffing-boxes at the surface, so that the pumps can be lowered the length of a steam-pipe without breaking the joints As the shaft is deepened the pumps are lowered by the capstan, and new pipes added at the top.

When the limit of the lift is reached, cisterns are placed in the shafts, and the pumping done in two lifts. The suspension-ropes are clamped at the pit-top, and coupled to the ropes on the capstan drums when it becomes necessary to raise or lower the pumps.

At Cadeby, over 400,000 gallons per hour were pumped by this system, from a depth of about 130 yards, eight "Denaby" sinking-pumps being employed.

One drawback to this method of pumping from sinking pits is the very heavy consumption of steam. It is impossible to keep the steam-pipes properly covered, and as the shafts are

* *Trans. I.M.E.*, vol. iii. p. 518.

necessarily wet, the loss of steam by condensation is excessive.

Economy of steam is not of paramount importance in temporary work such as sinking, and the advantages possessed by this method of pumping are so great as to render its universal adoption, in preference to hanging lifts, almost certain.

Pulsometers (see Chapter XXV.).—These pumps are frequently employed in sinking. They pump large volumes of water, and are in no way affected by sediment or even small stones; moreover, they are very light and compact, condense their own steam, and, having no moving parts except the valves, require but little attention. The serious drawback to the use of pulsometers, in sinking pits, is the limited height to which they can raise the water. With ordinary steam-pressures this is about 30 yards, so that several lifts are required when the feeders extend to a considerable depth. Pulsometers are very useful for pumping from the bottom of a sinking pit to a tank, from which a ram-pump delivers the water to the surface. They can be lowered, to follow the sinkers down, much more easily than an ordinary pump, and, when they are used in this manner, the main pump has only to be lowered once for each 20 to 30 yards that is sunk.

Electric Sinking-pumps.—Pumps driven by electricity are now being used for sinking. A strong steel frame carries a motor, which drives three single-acting ram-pumps through gearing, the whole being arranged to work when suspended, and take up as little room as possible in the shaft.

The motor is enclosed in an air and watertight casing, and the cables, from the dynamo at bank, are wound on a reel, and can be let out as required. These pumps are much heavier than steam-pumps of equal capacity, but the cables are lighter and more easy to deal with than the steam and exhaust pipes which they replace. There is also a saving in steam consumption as compared with steam sinking-pumps, for the latter work under the worst possible conditions as regards economy.

Special Systems of Sinking.—These special systems

of sinking are mostly of Continental origin. Until recently the shafts sunk in Great Britain (with several notable exceptions) have had no unusual difficulties to encounter; but in France and Belgium many shafts have had to be sunk through ground of such a character as to render special systems of sinking absolutely necessary.

The Kind-Chaudron System.—In this method of sinking the shafts are *bored* by percussion, and no water has to be pumped. In the original arrangement, the shaft was bored in two operations, a hole of small diameter (6 to 8 feet) being bored first, and afterwards enlarged. The boring tools, known as *trépans*, were lifted and dropped by a beam engine, and turned through a small angle at each blow, by men working on a scaffold at the pit-top. As the small shaft was bored, it was cleaned out by means of a large cylindrical sludger, similar to that used in the ordinary process of boring, which has already been described.

When enlarging the shaft to its full diameter, a tank was suspended in the smaller hole, to catch the *débris* made during the enlargement, the tank being raised when full by an engine, and emptied. The tubbing was made in cylindrical rings, having no vertical joints. As these rings were too large to be carried by rail, they had to be cast in a foundry erected close to the shafts. The rings were bolted together at the surface, and lowered down the pit, after it had been sunk through the water-bearing strata, into impervious ground. Before lowering the tubbing, a bed was prepared to receive it, this being done by a special tool attached to the bore-rods. A watertight joint at the bottom of the tubbing was secured by fitting the two bottom rings with flanges turned outwards, the lower ring being made small enough to allow the second ring to slide upon it. Moss was introduced between these two lower flanges, and as the lowest ring reached the curb-bed, it rested upon it, whilst the others slid down and compressed the moss by their enormous weight.

The Pattsberg Method of Sinking.—This method is now being employed (1903) in sinking two shafts at the Rhein

Preussen Collieries. It is similar to the Kind-Chaudron method in principle, but differs in very important details. The tubbing is forced down by hydraulic rams as the shaft is being sunk, and not lowered down afterwards as in the Kind-Chaudron method. The shaft is bored in one operation, and the *débris* is raised to the surface by a "mammoth" pump, no sludger being required. The mammoth pump consists of a pipe about 6 inches in internal diameter, extending from the bottom of the cutter to the surface; alongside this is another

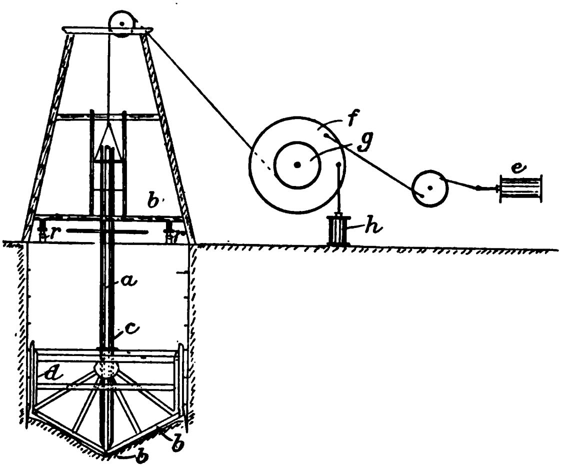


FIG. 54.—Pattsberg method of sinking.

pipe, $1\frac{1}{8}$ inch in diameter. Air at a very high pressure is forced down the smaller tube and ascends the larger one, carrying the *débris*—which is ground into mud—with it.

The general arrangement of this method of sinking will be understood from Fig. 54.

The cutter is carried by the bore-rod *a*, which is a pipe, about 6 inches in diameter. Water, at a pressure of 1000 lbs. per square inch is forced down this pipe, and issues from the holes *b*, *b* in the bottom of the cutter blades. This water stirs the sediment into mud, and enables it to be lifted to the

surface by the mammoth pump *c*. *d* is a guide-frame, working in guides fixed to the lower rings of the tubbing. Reciprocatory motion is obtained by the engine *e*, the disc *f* being driven by it through a crank and connecting-rod. The disc *f* is clamped to the small drum *g*, part of the weight of the cutter and rods being balanced by the steam-cylinder *h*. The rope from the drum passes over a pulley in the headgear, and is coupled to the bore-rods, as shown, guides being fixed in the headgear to steady the rods. As the pits deepens, the drum is unclamped from the disc, and additional rope run off. A rotary movement is given to the tool by men, through a turning lever. To raise the rods when the cutters have to be changed, the drum is unclamped, and turned by means of a worm, not shown in sketch. The tubbing is forced down by the hydraulic rams *r*, *r*; if it sticks fast, smaller rings of tubbing have to be forced down inside.

At the Rhein Preussen sinking, the cutter which was finally adopted was 19 feet in diameter, and weighed 12 tons; from 60 to 70 strokes per minute were made, the length of stroke being from 8 to 12 inches. The cutter-blades were changed about every fortnight, the operation taking about 12 hours. The rate of boring was from 2 feet 8 inches up to as much as 20 feet in one day.

Deepening Shafts.—It is frequently necessary to deepen a shaft whilst the upper part is in use for winding. When the winding shaft is only worked one shift, and there is no necessity for speed in sinking, the shaft may be deepened during the night. This is done by fixing up a sliding lorry at the pit bottom, in such a manner that the winding is not impeded; a length of rope is kept in the shaft, and coupled on to one of the winding ropes when sinking is in progress. After winding is finished for the day, the flat sheets which are in the way of the lorry are moved, and the loose rope is coupled up to one of the winding ropes, by passing it through a hole in the cage bottom. The sinking *débris* is not wound to the surface direct, but tipped into corves at the landing,

and either gobbed in the workings, or sent out on the cages.

When sinking and winding have to be done simultaneously, a separate engine has to be provided for the sinking, and the *débris* wound to a level below the landing of the cages. Fig. 55 shows an arrangement recently adopted at a Yorkshire

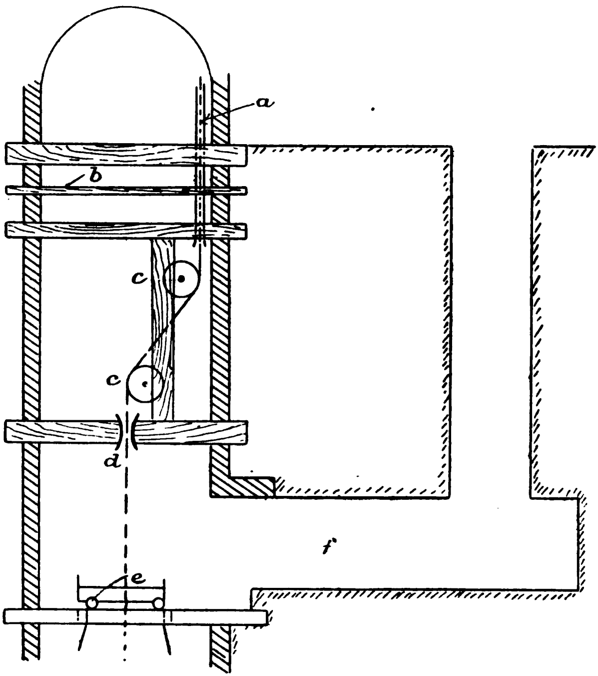


FIG. 55.—Deepening shafts.

colliery, which proved safe, efficient, and economical. The sinking-engines were placed on the surface, the rope being taken through the wrought-iron pipe *a*, fixed close to the shaft side, to be out of the way of the cages. A strong timber frame, *b*, was built into the shaft sides, just below the landing, and a bed of puddle rammed on its top, to keep back any

water made in the shaft. The winding-rope was led into the centre of the sinking pit by the guide pulleys *c*, *c*. *d* is the plate for the detaching hook, and *e* the rolling bridge. A short length of level road, *f*, was driven in the stone to make stand room for the corves, into which the *débris* was emptied from the hoppit. The corves, when full, were drawn up a staple pit to the winding-pit bottom landing, by a small steam-winch.

Frequently a few yards of solid strata are left in between the bottom of the winding pit and the top of the extension, a borehole being put through to ascertain the exact centre of the shaft, and for the passage of the rope. The object of this is to guard against injury to the sinkers in case a cage broke loose and fell down the shaft. An inclined plane is sometimes driven from the landing instead of a level and staple pit; and often the sinking engines are fixed at the top of the extension instead of at the surface.

Widening Shafts.—This is a very troublesome operation when the shaft has to be used during the process of widening either for winding or for ventilation. The widened shaft is usually made concentric with the original one, an equal area being blown out all round. The difficulties of widening shafts are greatly increased by the presence of pipes, conductors, or cables, which must not be disturbed by the blasting; and sometimes it is found to be more economical to take the whole width of the increase from one side.

When the shaft is not in use, a scaffold is hung some distance down it, and ashes are tipped on to it till they reach the surface. The men stand on these ashes whilst at work, and send them out as the widening proceeds. When the scaffold is reached, it is lowered another length, and ashes tipped on it again until the bottom of the widening is reached. This method is found to be both quicker and safer than the employment of a suspended scaffold.

Sinking Upwards.—Occasionally shafts have to be

sunk for a short distance from the bottom upwards; this occurs more frequently in metal than in coal mines. The usual method is to divide the shaft by a strong brattice; the space on one side is kept filled with the stones from the excavation, and forms a platform for the men to work on. After the shaft is completed, the brattice is removed, piece by piece, beginning at the top, the stones being thrown down at the same time.

Sometimes shafts are divided into three parts by two brattices, the middle compartment being kept full, and used as a platform, and the two outer ones forming the intake and return airways.

Sinking Contracts.—Shaft-sinking is usually let by contract, the contractor being paid a fixed sum per yard. If very much water is present, the risk may be too great to admit of a contract, and the sinking has to be done by day work. The following is an epitome of a recent sinking contract :—

1. The shaft to be sunk from the surface to the coal—an estimated depth of 700 yards.

2. The inside diameter, after the brickwork is put in, to be 20 feet; the shaft to be of an exact circle, and sunk perfectly plumb by the use of a centre line and ten side lines.

3. The shaft to be lined with brickwork 9 inches thick; any space between the back of the brickwork and shaft sides to be filled in solid with ashes or brickwork as required.

4. Curbs to be set perfectly level, notice to be given to the colliery engine-wright, so that he may examine and test them.

5. No shots to be placed within 1 foot of the sides, except by special consent.

6. The contractor to secure the shaft sides with timber and rings, and carry out the provisions of the Coal Mines Regulation Act.

7. The shaft to be sunk as quickly as possible, the men working day and night continuously, with never less than ten sinkers and a chargeman at work in the shaft.

8. The work to be done to the satisfaction of the colliery engineer, and in the most approved manner.

9. The contractor to find banksmen, and deliver the sinking material into wagons, after which the company will deal with it.

10. The company to find all materials except those afterwards mentioned; contractor to return the same in good condition.

11. The contractor to lade out a reasonable quantity of water; but if more than 1000 gallons per hour is made, the company are to provide a pump.

12. The contractor to provide all explosives, fuse and detonators, as well as all tools used by the sinkers.

13. No money to be paid to the contractor without a certificate from the company's engineer.

14. The contractor to supply the company with a sample and correct measurement of each stratum passed through.

15. 10 per cent. of the money due to the contractor to be kept back until the completion of the work.

The following are a few contract prices paid for sinking various shafts in the Midlands:—

Shafts, 18 feet in diameter, sunk in 1890,	per yard	£8	0	0
" 14 " " " 1897	"	£6	15	0
" 13 " " " 1900	"	£8	15	0
" extra for 9-inch brickwork	"	£1	10	0
" 16 feet in diameter, sunk in 1899	"	£11	10	0

CHAPTER VIII.

OPENING OUT.

Methods of Opening Out.—When the seams to be worked lie at a moderate gradient, the main roads are made in the coal; but if the seams are very steep they are usually won by level drifts driven across the measures, as shown in Fig. 56. The shafts are first sunk to *a*, and the rise coal in the various seams cut by the drift worked uphill. As these become exhausted, the shaft is deepened to *b*, and other drifts driven to win the strip of coal lying between *a* and *b*, and so on.

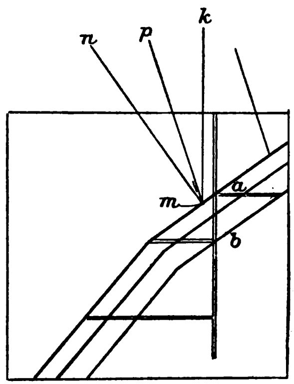


FIG. 56.—Opening out in steep seams.

Pit Bottom.—The mouthings at the pit bottom are secured by arches, the shaft being belled out to obtain the necessary width. Often pit bottoms are made very wide—30 feet and upwards—and sidings are provided for both full and empty corves, side by side. These wide pit bottoms necessitate very costly masonry and extensive excavations; and when empty and full corves run side by side, it is difficult to grade the roads and control the traffic. The best arrangement for a pit bottom is to have some roads entirely for full and others for

empty corves; all the traffic has then a definite direction, and can be automatically controlled. As much siding room as possible should be provided, so that the winding engines can be kept at work for an hour or two, in case the haulage should fail, and also to ensure a good start in the mornings. Some of the modern collieries have siding room for four or five hundred tons in the pit bottom.

The cages at large collieries always have several decks, and arrangements should be made for loading them simultaneously. There are several methods of doing this: one very common plan, when double-decked cages are employed, is to have two landings, one level with the upper and the other with the lower deck; the corves from one district serve the upper, and those from another district the lower deck. It is preferable to arrange the roads so that the corves from any district can be sent to either deck.

Fig. 57 shows a simple method of loading three decks simultaneously. The whole of the full corves are brought along the road marked *ab*, which is level with the top deck; the corves for the second and third decks are lowered by the drop-cages *c* and *e* respectively. Each drop-cage is connected to another small cage on the other side of the shaft, so that the weight of the descending full corves raises the empties to the main level. Each cage is served by its own road, which is used solely for it. The cages for the empty corves are made heavier than those for the full ones, so that when both cages are empty they run back to their proper landings ready for the next load.

The roads for the full corves should dip towards the shaft, and the roads for the empty ones from it. This means that the empties are delivered at a level considerably below the main haulage road, and mechanical arrangements must be provided to raise them. A gradient of 1 in 60 is usually found to work well at a pit bottom.

Shaft Pillars.—When coal is worked at a moderate depth from under any considerable area, the surface invariably sinks,

the amount of subsidence being from 50 to 70 per cent. of the thickness of the seam, so that if a seam 6 feet in thickness is worked, the level of the surface above it will sink between 3 or 4 feet.

Subsidence always extends beyond the excavation, and the area subject to this "pull" varies with the depth of the seam; hence, when pillars are left to protect shafts or buildings, they

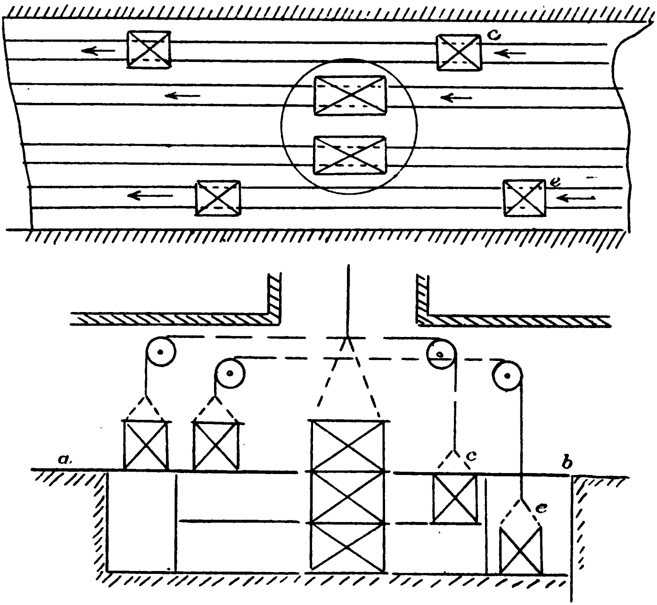


FIG. 57.—Method of simultaneously changing the corves on three decks.

must be larger for deep seams than for shallow ones. It is of the utmost importance to protect shafts from the effects of subsidence, and pillars have to be left for this purpose. If these pillars are too small the shafts may be crushed and forced out of plumb, and if they are tubbed, the tubbing may be fractured with disastrous results; moreover, the roads through the pillars will be difficult to maintain. There are

several rules for determining the size of shaft pillars, but the pillars provided by most of them are quite inadequate.

The following are examples of shaft pillars left in collieries recently sunk, all working coal by the longwall system.

1.	Depth,	750	yds.,	thickness	of	seam,	7	ft.,	diam.	of	pillar,	600	yds.
2.	"	540	"	"	"	"	6½	"	"	"	"	520	"
3.	"	240	"	"	"	"	5	"	"	"	"	250	"
4.	"	500	"	"	"	"	4¼	"	"	"	"	500	"

The general practice is to leave rectangular pillars having their sides equal in length to from two-thirds to the full depth of the shaft: thinner seams, of course, require less pillars than thicker ones. Theoretically, a shaft pillar in a level seam should be circular, but in practice it is found more convenient to make them rectangular.

A pillar, 700 yards square, has an area of over 100 acres. This may appear very large, but it must be remembered that the shaft pillar has to support the engine houses, chimneys, etc., as well as the shaft.

At one time it was thought that the line of fracture or break, resulting from coal workings, ran vertically to the surface, as shown at *km*, Fig. 56. This was found to be incorrect when the seam was inclined, as also was the theory that the line of fracture runs at right angles to the dip of the seam, as at *nm*. It is now generally recognized that the break runs about midway between these two lines, as *pm*, Fig. 56. This shows that in inclined seams it is necessary to arrange the pillars in such a manner that more coal is left to the rise of the object to be supported than to the dip, in order to support the object equally on all sides.

Water Levels.—When much water is made, water-levels have to be driven to form standage for the pumps and to catch the rise water and prevent it following the workings down as the coal is got to the dip. Water-levels should be driven on the dip side of the main roads, so as to drain them; and if it can be conveniently arranged, the top of the water-level should be lower than the floor of the main level, otherwise the water

will flow from it into the main level before it is full. For instance, if the seam dips 1 in 10, and the water-levels are 5 feet high, the distance between the main and water levels should not be less than $5 \times 10 = 50$ feet. When water-levels are used as standage for pumps, they must be driven dead level, or they will overflow at their lower end before the upper end is full, and so will not be able to contain an amount of water equal to their full capacity. When a water-level meets a fault, the level must be turned either to the rise or to the dip to catch the coal at the same level on the other side of the fault. If the fault is a downthrow, the level must be turned to the rise until it cuts the coal, and if an upthrow, it must be turned to the dip. Fig. 58 shows a plan and section to illustrate the method of crossing a fault with a water-level. In the section *ab* shows the position of the seam above the fault, and *cd* its position below the fault. If the level *g* were continued in a straight line it would have to dip to regain the seam, but by turning it to the rise, as shown, the coal can be won and the original level maintained. All important pumping engines should have sufficient lodge room to hold at least a couple of days' water. If the feeder amounted to 400 gallons per minute, and the water-levels were 6 feet 6 inches wide and 5 feet high, the length of levels necessary to hold the water made in 48 hours would be 5671 feet. This is calculated as follows: The number of gallons made in 48 hours is $400 \times 60 \times 48$, and, as there are $6\frac{1}{4}$ gallons in one cubic foot, the number ot gallons divided by $6\frac{1}{4}$ gives the cubic feet of space required. The area of the road is $6\frac{1}{2} \times 5$ feet, so that the required length

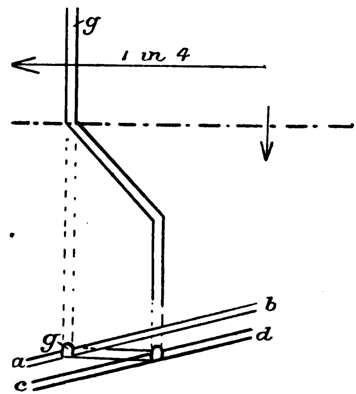


FIG. 58.—Water-level crossing fault.

is the number of cubic feet divided by the area, or $\frac{400 \times 60 \times 48}{6\frac{1}{4} \times 6\frac{1}{2} \times 5}$ $= 5671$ feet.

For a large quantity like the above, it would be necessary to drive several parallel levels connected by slits.

Direction of Main Roads.—The direction of the main roads depends upon the dip of the seam, the position of any known faults, and the shape of the royalty; it is also to some extent governed by the system of working the coal, and the method of haulage. The output of some of our large collieries reaches from 250 to 300 tons per hour, and to deal with this large quantity several independent main roads are absolutely necessary. They should be carefully laid out from the commencement of the colliery, so as to divide the whole area of the royalty among them: and as far as possible they should be perfectly straight. The branch roads should leave the main roads by regular curves of large radii. In longwall work the gates should be systematically cut off by cross gates or levels. The direction of the gates depends upon the dip of the seam, and upon the line of the faces, which, in its turn, is usually governed by the cleavage lines of the coal. Seeing that every yard of road may be expected to cost something per year for maintenance, it is obviously desirable to keep down the length of the roads as much as possible, but, on the other hand, it is a very serious evil to have too little pit room, leading, as it does, to a diminished output.

After the direction of any road is decided, it is necessary to fix lines or marks in the road, for the guidance of the workmen. Lines are usually hung 6 or 8 feet apart. A greater distance between them would be preferable, but if they are much further apart they cannot be illuminated by one lamp, and two men have to be employed to take a sight. To hang a pair of lines, a dial is set up, and its sights clamped at the correct angle as read off from the magnetic needle.

The approximate position of each line is then marked on the roof, by observations taken through the dial sights, and

holes are drilled a few inches into the roof. Soft wooden plugs are driven tightly into these holes, and the pivots carrying the lines are knocked into them and very carefully adjusted to the exact line of the dial sights. To take sight, a lamp is held in the face and moved about to the instructions of the observer, who stands behind the lines and notes when they and the lamp are in exact alignment. A mark is then made on the roof over the lamp, and the series of marks made day by day in this manner keeps the workmen in the right direction.

Lines are often hung from the bars which support the roof. These, however, are apt to be moved by the pressure, which would throw the road wrong.

When the roof is good and even, chalk lines may be marked on it and carried forward as the road advances. To mark the line, the ends of a chalked cord are held tightly in the correct position, and its centre is pulled down; this causes the cord, when released, to rebound smartly, leaving a chalk mark on the roof.

Roads driven in the coal have to follow the inclination of the seam, but stone drifts driven across the measures must be arranged with a fixed gradient. This gradient is maintained by means of a templet, which consists of a spirit-level or plumb-bob mounted on a straight-edge. A tapered strip of wood is nailed to the bottom of the straight-edge, the amount of the taper being equal to the required inclination of the drift. If the road had to dip or rise 1 in 6, and the straight-edge were 4 feet long, the tapered strip nailed to the bottom would be 8 inches deep at one end, and come to a point at the other. To test the gradient, the templet is placed on one of the rails of the drift, one end of which is raised or lowered until the bubble of the spirit-level is in the centre of its run.

When a road is to be used for horse haulage alone, it is more important to keep it level than straight, and it is usual to maintain a favourable gradient by slightly altering the direction of the road to suit the undulations of the strata. Roads used for mechanical haulage, however, should always be kept straight, and undulations disregarded.

It frequently happens that cross-measure drifts have to be driven, either to win coal cut off by faults, or to connect two seams for haulage or ventilation. The length of such drifts may be calculated as follows:—

A seam dipping 1 in 5 is thrown 20 yards down by a fault, what length will a drift dipping 1 in 3 be to win the coal on the other side of the fault?

Dip of drift is 1 in 3 = 5 in 15
" seam " 1 in 5 = 3 in 15

So that the drift gains 2 yards vertically upon the seam for every 15 yards measured *horizontally*, hence the 20 yards will be gained in $\frac{15 \times 20}{2} = 150$ yards.

To calculate the length of the drift measured on the incline,

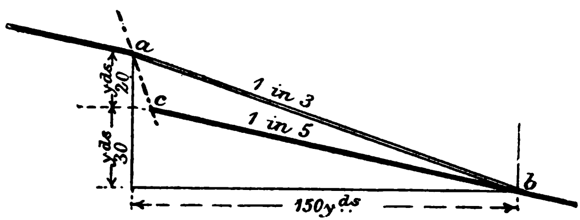


FIG. 59.

the total fall must first be found. As the seam dips at the rate of 1 in 5, the fall in a horizontal distance of 150 yards is $\frac{150}{5}$ = 30 yards, hence the total fall of the drift is $20 + 30 = 50$ yards.

The actual length of the drift is equal to the hypotenuse of a right-angled triangle, having base and perpendicular 150 and 50 yards respectively, and $\sqrt{150^2 + 50^2} + 158{\cdot}1$ yards. The method by which this is arrived at will be understood by references to Fig 59, in which *ab* represents the drift, and *bc* the seam.

CHAPTER IX.

MINERS' TOOLS.

Picks.—These are the most widely used of all the tools employed in mining; they vary considerably in shape and weight, and are employed for a variety of purposes. For coal holing or cutting, the blades are from 1½ to 3 lbs. in weight, and about 15 inches long; the shafts or helves being of ash or hickory, and about 2 feet 6 inches long. The blades are straight, or very slightly curved; they should be made of steel throughout, and the ends should be square in section, and taper gradually to the point. The helves may be fixed to the blade; but "interchangeable" picks, in which the blades are loose and can be changed, are the most common, as during a shift a workman may "mar" several blades, and these interchangeable picks greatly lessen the weight he has to carry to and from his work. *a*, Fig. 60, shows the "Universal pick," made by the Hardy Patent Pick Company, in which the helve is slipped through the eye of the blade and secured by becoming wedged on the ferrule.

The picks used in stone are much heavier than those used in coal. They are from 4 to 7 lbs. in weight, and the helves are made both longer and stronger; the blades are square or octangular in section, and made thick almost up to the points.

Dressers.—*b*, Fig. 60, shows a dresser. They are employed for breaking up, or pulling down, large masses of coal or stone; and are also used by plate-layers. One side of the head forms a hammer, and the other a curved pick. As they are used almost entirely for wrenching, the helve is curved, and strengthened by iron bands where it fits into the eye.

Wedges.—These are used for breaking down coal or stone, and for cutting up large pieces after they have fallen. The coal wedge, *c*, Fig. 60, is 8 to 12 inches long, and about a couple of inches wide, and 1½ inch thick at its head, with a flat chisel point. For use in stone, the wedges are smaller; they are similar in shape to the coal wedges, but come to a point at the end.

Hammers.—These vary very greatly in size and weight, in accordance with the work for which they are employed. They should be made of steel, tapering slightly from the eye, and

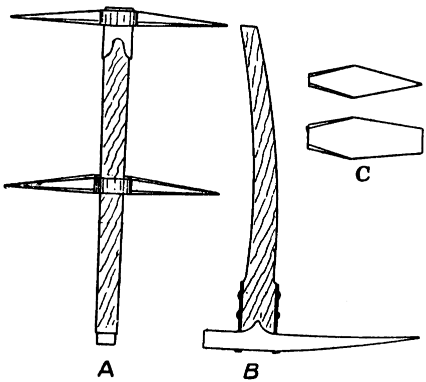


FIG. 60.—A, "Universal" pick; B, dresser; C, wedge.

champered off towards the faces. The weight of the head is from about 5 lbs. for use in coal, to about 10 lbs. for stone.

Shovels.—The plates are of steel, slightly turned up at the edges, and coming to a point at the ends, to enable them to easily enter loose heaps of stuff. The centre of the plate is strengthened by being bent into a crease, terminating in the straps by which it is attached to the helve. The helves or handles are about 2 feet 6 inches long, and are set at an angle of about 150° to the plates; they are circular in section, so as to be comfortably handled, and terminate in a crutch or box-handle. The size of the plates vary from 10 inches wide, when

used in stone, to 16 inches in width, when used for shovelling coal.

Drills.—Shot-holes may be either bored by percussion, or by a rotary hand-boring machine. For coal or soft rock, "jumpers" are sometimes employed. These consist of a bar of iron 5 or 6 feet long, having a steel bit at one end, and sometimes an enlargement or bulb on the bar near the other. The workman grasps the bar near the bulb and works it backwards and forwards in the hole, making the bit strike the bottom of the hole at a different place by giving the bar a slight turn between each blow. This method is only adapted for soft material, and when the rock is hard, striking drills have to be employed. A set of striking drills consists of two or three drills of different lengths, the drills being made from octagonal bars of steel, forged into a chisel-shape at one end. A single-handed set consists of two light drills, one being about 20 and the other about 40 inches long. To use these drills, the miner holds the drill in one hand and strikes the top with a light hammer which he holds in the other, turning the drill through a small angle at every blow. The shorter drill is used at the commencement, and when the hole becomes too deep for it the long one is employed. Small holes only can be bored by single-handed drills, and they are not suitable for hard rocks. In the double-handed set there are three drills of about 20, 30, and 50 inches long respectively. One man holds the drill, raises it slightly, and turns it between each blow, whilst another—or, if the ground is very hard, two men—strike the top with heavy hammers. The bit of each drill is made rather smaller than the one it follows, to enable it to pass easily down the hole, care being taken to keep the holes circular and of even diameter all through.

Holes can be drilled by this method in extremely hard ground; in fact, the one advantage it possesses over the rotary method is that it will face harder rocks, and so can be applied in strata too hard for the rotary machines.

Striking drills are now nearly always made of steel throughout; formerly they had iron shafts and steel bits. Steel drills

are lighter than the iron ones, and steel transmits a blow better than iron.

Rotary Hand-boring Machines.—For ordinary colliery work, rotary machine drills have almost displaced striking drills: they are quicker, bore a hole with less labour, and in ordinary ground only require one man to work them. A very good type of machine is shown in Fig. 61, which is an illustration of the "Elliott" drill made by the Hardy Patent Pick Company. It consists of a drill or auger (*a*), worm (*b*), standard (*c*), ratchet and handles (*d*), and worm-wheel (*e*). The drills are made of flat bar steel, twisted into a spiral; from two to four of increasing lengths forming a set. The worm or screw is provided with a socket at one end, into which the drill is fitted; the other end is forged square to receive the ratchet. The pitch of the screw is about half an inch—that is, the screw advances at the rate of half an inch for every revolution. The screw works on a worm-wheel, having teeth cut in its outer edge. This worm-wheel is carried by a split ring provided with hinges and tightening screw. When the tightening screw is quite slack, the drill, when turned, does not advance, but moves the worm-wheel round in the split ring, so that when the drill is working, sufficient friction has to be applied to the split ring by the tightening screw to prevent the wheel from turning with ordinary pressure. If extra pressure be applied, the wheel moves, instead of the drill advancing to the full extent of the pitch of the screw. The effect of this is to make the feed partly automatic and prevent breakages, as when, owing to the hardness of the ground, extra pressure is applied, part of the strain and motion is taken by the worm-wheel.

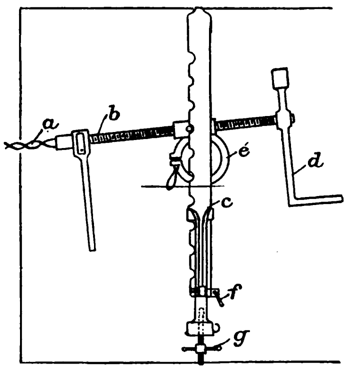


FIG. 61.—Elliott drilling machine.

The machine is provided with two ratchets, one being fitted to each end of the screw; by the use of ratchets the to-and-fro movement of the handles gives a rotary movement in one direction to the screw and drill. The end ratchet can also be used as a crank handle when required.

Ratchets are slower than cranks, but more power can be applied through them, and they can be used when a hole is being drilled too near the roof to allow a crank to be turned. The standard is made telescopic, as shown in Fig. 61. It consists of a double frame of steel, having notches to carry the pin, attached to the sleeve of the split ring. Rough adjustment for height is made by drawing down the outer frame and securing it in a position by the screw *f*; after this is done the machine is fixed in position, and tightened up by the bottom screw *g*.

To use the Elliott or similar machine: a hole a few inches in depth is first stamped into the coal or stone with a pick; the machine is then erected and tightened up, the correct distance from the face being obtained by a rough measurement.

The shortest drill is fitted into the socket, and the ratchet is worked until the screw has advanced its full length. The tightening screw of the split ring is then slackened, and the screw *b* is pushed right back, turning the worm-wheel as it moves. A longer drill is then fitted into the socket, and the operation repeated until the hole has reached the required depth.

When the rock is very strong both ratchets are used, but for soft ground only one is required. A simple form of drill, which is much used in stone, consists of a worm working through a nut in a barrel; no standard is employed, the barrel being set against a prop. The screw thread is of small pitch, so that the drill advances slowly, but will penetrate hard rock.

Sharpening and tempering Tool Steel.—Drills and picks are sharpened by being heated to redness and hammered on an anvil. After they are sharpened they must

be hardened and tempered. This is an operation which requires great skill and care; different kinds of rock require tools of different tempers, and the smith has to make the degrees of hardness suitable for the various places in the pit. If steel is heated and allowed to cool gradually, it becomes soft; but if the temperature is reduced very rapidly it becomes very hard and brittle. After the tool has been sharpened it is reheated, and as it cools it shows various colours, each of which represents a different degree of hardness.

The smith cools the point slightly by water, after the reheating, and watches the play of colour; and as soon as the right one is reached, he cools it rapidly by holding it in the water. The colours vary from light yellow to brown, purple, violet, and dark blue.

In tempering steel care must be taken not to burn it by applying too great heat; and the sharpening should be done by striking the point lightly and quickly until it is quite black. When hardening steel picks, the end only should be heated up to about 1 inch from the point; and when cooled off, the water should not reach more than about $\frac{3}{4}$ inch from the point.

Blasting Tackle.—As shot-holes are being drilled the *débris* made by the cutting tools must be cleaned out from time to time. Drills of the rotary class are to some extent self-cleaning, but all holes require carefully cleaning out before the explosive is inserted. A copper or composition *scraper* is used for this purpose. It consists of a rod 4 or 5 feet long and from $\frac{1}{4}$ to $\frac{3}{8}$ inch in diameter; one end is shaped into a handle, and the other is hammered flat and bent back at right angles to form a scraper, by which the sludge or dust is drawn from the hole. The *rammer*, or *stemmer*, is a copper or wooden rod, swelling out at one end to rather less than the diameter of the hole, and having a small groove along the side for the passage of the fuse or needle.

The *needle*, or *pricker*, is a thin copper rod, having a handle at one end, and gradually tapering to a point at the other. It is only required when shots are fired by "germans" or

"squibs," and not by fuses. The process of firing a shot by a squib is as follows:—

After the hole has been cleaned, the end of the needle is forced into the blasting powder cartridge, and is pushed carefully into the end of the hole. The hole is then stemmed by ramming it tightly with clay or other suitable material by means of the stemmer. This is done until the hole is stemmed to the end, when the needle is carefully withdrawn, leaving a small hole right through the stemming to the charge. A squib, which is a straw, or paper tube filled with powder, having a piece of touch-paper at one end, is then fixed in the mouth of the hole. When the touch-paper is lighted it sets fire to the powder, and a train of sparks is projected along the small hole left by the needle, and fires the charge. This method of firing shots is now becoming obsolete.

Ringer and Chain.—For drawing timber from goaves, a ringer or dog and chain must be used. This, in its simplest form, is an iron or steel bar from 3 feet 6 inches to 4 feet long, with one end bent into a claw, about 6 inches from which a length of chain is attached. The chain is secured to the prop which has to be drawn, and the end of the claw fixed against another prop, which must be sufficiently firm to bear the strain.

The leverage is 6 or 8 to 1, so that if a pull of 1 cwt. is applied to the long end of the bar, the strain on the prop and chain will amount to 6 or 8 cwts.

Sylvester's Patent Prop Withdrawer.—This implement, which is a great improvement on the ordinary dog and chain, is shown in Fig. 62.

a is a notched steel bar, 3 feet long, 1½ inch deep, and ⅝ inch thick, the notches are 1 inch apart and about half an inch deep. At one end of this bar is a swivel carrying a light chain, *b*, and at the other end a stop, to prevent the sliding block *c* moving too far. The sliding block is propelled along the bar by means of the lever *d*, the short end of which engages with the notches as shown. The back of the block is provided with a recess, shaped so as to grip any link in the

chain *e*. The block is kept from sliding backwards by means of the catch-bolt *g*, which can be drawn clear of the notches when required. To draw a prop with this appliance, the chain *b* is secured to a prop or other firm object and the longer chain, is lashed round the prop which is to be withdrawn. *e* is then pulled tight, and one of the links slipped into the recess in the block; the lever is moved backwards and forwards, drawing the block, chain, and prop along the bar towards the fixed prop. The block can be freed and slipped back to the end of the bar by drawing the catch-bolt *g* clear of the notches. The advantages of this apparatus over the ordinary ringer and chain are that the leverage is very much greater, being 30 to 1

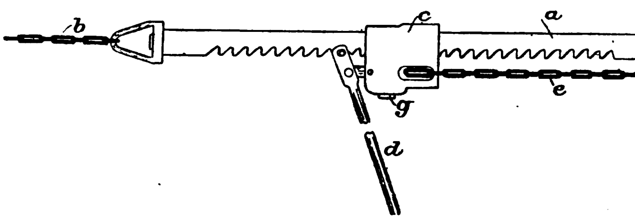


FIG. 62.—Sylvester's patent prop withdrawer.

as against 7 to 1, and that the strain is not released when the weight is taken off the lever. The use of these prop withdrawers is not confined to pulling out props; they may be employed with advantage for drawing the ends of ropes together when they have to be spliced, separating corves which have got off the road and become jammed, and for a variety of purposes which require the moving of heavy loads.

Mechanical Wedges.—Many mechanical appliances have been patented for breaking down coal or stone in order to avoid the use of explosives.

As yet none of these machines have been widely adopted, though some do very good work when the conditions are favourable. They frequently fail when used in soft tough coals, as the result of the expansion is to crush and grind up the coal round the hole in which the wedge is inserted, instead of rending the coal and bringing it down.

Fig. 63 shows the *Hardy Patent Pick Company's Multiple Wedge.*—These wedges are made in sizes varying from 18 inches in length and 1½ inch in diameter to 4 feet in length and 2 inches in diameter. To use them, a hole is first bored in a similar position to that which would be necessary if an explosive were to be used; the depth of the hole should be about 6 inches more than the length of the wedge, and its diameter should exceed that of the wedge by about ⅛ inch. The feathers *a* are first placed in position, and the split-wedge

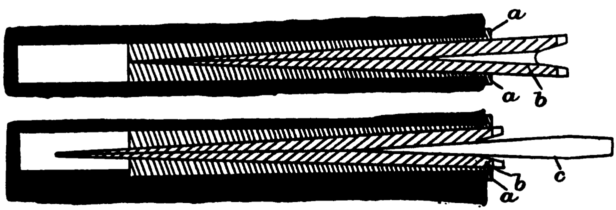


FIG. 63.—Multiple wedge.

b driven up between them. If the expansion is not sufficient to bring down the stone or coal, the other wedge, *c*, is driven between the halves of the split wedge *b*, and further expansion is obtained. This apparatus is lighter and cheaper than most mechanical wedges, and requires a much smaller hole.

Hydraulic wedges are sometimes employed; in some forms the rending action is accomplished by drawing long wedges between pairs of feathers, whilst in others small rams are forced against the upper side of the hole.

CHAPTER X.

EXPLOSIVES.

Explosives.—An explosive is a substance which contains within itself all the ingredients necessary for complete and rapid combustion, the gases resulting from such combustion occupying a much greater volume than the explosive itself. The action of an explosive is simply this: The charge is lit or detonated, and is instantly changed to gases; this causes it to expand with extreme rapidity, and, if there is not ample room for expansion, great pressure is generated.

The power of an explosive depends upon the quantity and density of the gases produced, and the rapidity of its action upon the speed with which the explosive is changed into gas. Some explosives are much quicker in action than others. For example, if blasting powder be placed on a stone and fired, the stone will be undamaged, because the change from solid to gas takes place comparatively slowly, and the expansion relieves itself in the air. But if dynamite be fired in the same manner, the stone will be shattered, because the expansion takes place so rapidly that it has not time to spend itself entirely in the air, the action being more in the nature of a blow than of gradual pressure.

The chemical change which takes place at an explosion is accompanied with great heat, which adds to the expansion of the gases. An explosion may be regarded as extremely rapid combustion, as when any substance burns, gases are generated, although the action may be very slow. An ordinary combustible requires a supply of oxygen from external sources,

usually the air, but an explosive contains oxygen usually in the form of a nitrate.

Explosives are employed in mining for breaking down coal, ripping, etc., and for driving stone drifts, and sinking When used in coal the explosive should be slow in action, so as not to shatter the coal, and should "spread" well, so as to bring down a large area; the same qualities are required for ripping and similar work. For driving stone drifts, and sinking in hard ground, a high explosive which is quick in action is usually preferred, one advantage being that a smaller hole is required.

Legislation.—The great dangers resulting from the use of explosives in dusty or gaseous mines (see Chapter XXVIII.) are now fully recognized; and the "Explosives in Coal Mines Order of 1899," with subsequent modifications, has been drawn up to meet them. The chief provisions of this Order are—

1. (1) "Permitted" explosives only must be used in all mines in which a dangerous quantity of gas has been found within the previous three months.

(2) "Permitted" explosives only must be used in all roads and in every dry and dusty part of any mine which is not wet throughout.

(3) In all such coal-mines as mentioned above the use of "permitted" explosives is absolutely prohibited, unless the following conditions are observed :—

(*a*) Every charge of explosive must be placed in a properly drilled shot-hole, and be properly stemmed.

(*b*) Every shot to be fired by electricity, or by some other means equally secure against the ignition of gas or dust.

(*c*) Every charge shall be fired by a competent person appointed in writing, and not being a person whose wages depend upon the amount of mineral to be gotten.

(*d*) Each explosive to be used in the manner prescribed in the schedule.

3. All explosives are prohibited in the main haulage roads and intakes, unless all workmen have been removed from the seam, and all others communicating with the shaft at the same level, except the men engaged in firing the shot, and not more than ten other persons who may be employed in attending to engines, horses,

etc., or in inspecting the mine; or unless a permitted explosive is used, and every part of the roof, floor, and sides of the main haulage road or intake are thoroughly wet within a distance of 20 yards from the shot.

This section does not apply to such portions of the main haulage roads or intakes as are within 100 yards of the coal face. A "main haulage road" means any road which has been or is being used for moving trains by gravity or mechanical power.

Detonators are to be issued only to shot-firers, and must be kept by them in a locked case.

A list of "permitted" explosives is added as an appendix to the Order, giving composition, number or strength of detonator to be used with each, and method of firing.

Composition of Explosives.—The principal explosives used in coal-mines may be roughly divided as follows:—

(*a*) Gunpowder, and similar compounds.
(*b*) Nitro-glycerine compounds.
(*c*) Ammonium nitrate compounds.

Gunpowder is a mixture containing—

Saltpetre (potassium nitrate) from	65 to 75 per cent.
Charcoal	15 per cent.
Sulphur from	10 to 20 per cent.

For blasting in coal, there is no explosive equal to gunpowder. It is much slower in its action than the high explosives of the nitro-glycerine and ammonium nitrate class, and therefore brings down the coal in larger lumps, and makes much less slack. It also has the great advantage of not requiring a detonator to fire it, a simple fuse or squib being all that is necessary.

The use of blasting powder in coal-mines is now greatly limited by the "Explosives in Coal Mines Order," as it is not upon the permitted list.

Nitro-Glycerine Compounds.—Nitro-glycerine is a very powerful explosive, very rapid in its action; it is of high specific gravity, and is not affected by water. Nitro-glycerine

compounds are very suitable for blasting hard rock, as only small holes are required; and, owing to their density and plastic nature, they do not fill up much of the shot-hole, but lie in the bottom, and can be pressed down to fill up the whole of that portion of the shot-hole which they occupy.

Dynamite.—In its liquid state nitro-glycerine is dangerous and inconvenient to use, but it is commonly employed when mixed with some absorbent substance. Dynamite consists of nitro-glycerine absorbed by *kieselguhr*, which is a porous earth found in Hanover. The proportion of nitro-glycerine present depends upon the strength which is required, and varies up to as much as 75 per cent. (by weight) of the whole compound. In common with the other high explosives, dynamite must be fired with a detonator, and is liable to explode if subjected to a very severe shock or blow. It is not on the permitted list, but is much used for sinking, quarrying, etc.

The list of "permitted" explosives contains the names of several nitro-glycerine compounds, of which carbonite may be taken as an example.

Carbonite.—The authorized composition of this explosive is, for every hundred parts by weight of the finished explosive,—

Not more than 27 parts, or less than 25 parts of purified nitro-glycerine.
Not more than 36 parts, or less than 30 parts of nitrate of barium or nitrate of potassium.
Not more than 37 parts, or less than 34 parts of wood meal.
Not more than 5 parts, or less than 4 parts of moisture.
With or without not more than ½ part sulphuretted benzol.
With or without not more than ½ part carbonate of sodium and carbonate of calcium.

It must be used in a non-porous wrapper, and fired with a detonator of not less strength than that known as No. 6. If in a frozen condition, it must be thawed in a safe and suitable manner.

Nitro-glycerine freezes at about 46½ degrees Fahr., that is,

when the temperature is about 14½ degrees above the freezing-point of water; its compounds are dangerous when frozen, and should be thawed in a can having an outer case to contain warm water. In no case should cartridges be exposed to the direct heat of the fire.

The majority of the "permitted" explosives belong to the ammonium nitrate class. Ammonium nitrate is a very powerful explosive; it "spreads" more, and is not quite so quick in its action as nitro-glycerine, and is in consequence more suitable for blasting coal or ripping. Its compounds are light and bulky, which renders them unsuited for very hard ground. It does not freeze, but is affected by moisture, and must be used in watertight cases when the holes are wet.

Westfalite, No. 1.—This is a good example of a nitrate of ammonium explosive; its authorized composition is, for every hundred parts by weight,—

Not more than 96 parts, or less than 94 parts nitrate of ammonium.
Not more than 6 parts, or less than 4 parts rosin.
Not more than ½ part moisture.

The explosive must be used in (*a*) a wrapper of stout paper, thoroughly waterproofed with paraffin wax; or (*b*) a case made of an alloy of lead and tin, waterproofed with paraffin wax; or (*c*) a non-waterproofed wrapper of paper, the outer waterproofed paper having been previously removed.

The explosive is only to be used with a detonator of not less strength than that known as No. 7.

"*Safety*" *Explosives.*—A perfect explosive, for use in coal, should break down the coal in large masses without shattering it; should have a temperature of detonation so low as to render the firing of gas or coal-dust impossible; and should not give off poisonous or unpleasant fumes when it explodes. None of the explosives now on the market come quite up to this standard. All the high explosives shatter the coal more or less, none are absolutely safe when fired in a dangerous atmosphere, and the reek from all, though perhaps not dangerous,

is certainly unpleasant. All "permitted" explosives are much less liable to fire gas or dust than blasting powder, as the gases they produce tend to quench the flame and lower the temperature of detonation.

A committee of the North of England Institute of Mining Engineers, after making a series of tests of "flameless explosives," came to the following conclusions :—

All high explosives upon detonation produce evident flame, and are liable to ignite mixtures of air and fire-damp or coal-dust, though they are all safer than ordinary blasting powder. The proportion of coal-dust in air necessary to form an explosive mixture is much less than has hitherto been thought to be the case.

The risk of an explosion, when using high explosives, is only diminished, and not abolished.

Explosives alter in character if they are improperly kept. In view of the changes in the composition of an explosive which are made by the makers from time to time, the composition and date of manufacture should be printed on the wrapper of each cartridge.

Detonators.—High explosives have to be fired with *detonators*. These are hollow copper cylinders, closed at one end, and containing the detonating agent, usually a mixture of fulminate of mercury and chlorate of potash, which explodes with a strong local action when fired by a spark. It is very important that perfect detonation should be produced, otherwise the explosive may burn away slowly rather than explode, the result being a great or total loss of power, accompanied by unpleasant fumes. Some explosives (more especially those of the ammonium nitrate class) are very liable to incomplete detonation when damp, or improperly made. In some cases part of the explosive only is detonated, the remainder being compressed into a hard mass. To ensure perfect detonation, the detonator must contain sufficient fulminate, and the cylinder containing it must be strong enough to offer considerable resistance to the shock of its explosion. The detonator has a better chance of doing its work well when

placed at the back of the hole; or, if several cartridges are employed and fired by electricity, it may be placed in the middle one with advantage.

The explosives on the permitted list have to be fired with detonators varying in strength from Nos. 6 to 8. A No. 6 detonator contains 15 grains of a composition containing 80 per cent. of fulminate of mercury and 20 per cent. of chlorate of potassium. A No. 8 detonator contains 30·9 grains of a composition containing 80 per cent. of fulminate of mercury and 20 per cent. of chlorate of potassium.

Detonators deteriorate very seriously if allowed to become damp; even when they are sufficiently dry to explode they may be so much weakened as to fail to completely detonate the charge. When the detonator is exploded without firing the charge, the failure is usually attributed to a fault in the explosive, although it may be entirely owing to the detonator.

To fire a high explosive with an ordinary fuse: First cut off the end of the fuse obliquely, then carefully examine the detonator to see that it is quite clear of dust or dirt of any sort; next push the fuse into the open end of the detonator, and nip its edges on to the fuse to hold it firmly in position. Then make a hole in one of the cartridges to be fired, using a sharp wooden peg for the purpose, and push the detonator into the explosive in such a manner that it cannot be pulled out if reasonable care be employed. Next push the cartridge into the hole with a wooden or copper stemmer; stem very lightly at first with clay, and very heavily in the later stages. The shot is then fired by lighting the end of the fuse, after making certain that all men and ponies are out of the way of any flying pieces of stone or coal.

Detonators should be handled with the greatest care, otherwise they are extremely dangerous. If dust or fluff gets into the open end, it should be shaken or blown out, and on no account should it be removed by the insertion of a piece of wire or anything else. Every explosive gives the best results only when properly stemmed, although good stemming is not so necessary with the high explosives as with blasting powder.

Fuse is made in several qualities: only good fuse should be used, as misshots are very costly. Ordinary fuse burns at the rate of about 30 inches per minute. It is usually lighted by an open light, or Bickford's igniters may be used. These consist of tin cylinders, rather larger than ordinary detonators. They are open at one end, and the closed end contains the igniting mixture.

One of these igniters is slipped on to the end of the fuse which projects from the shot-hole, and held in place by being nipped at its open end. The closed end is then nipped with special pincers; the pressure on the mixture fires it, and this in its turn lights the fuse. It is desirable, when firing several shots simultaneously with safety fuse, that the lengths of fuse should vary, so that a misshot may be detected.

Shot-firing by Electricity.—In mines where "permitted" explosives are used, shots are usually fired by electricity. The advantages of electric-shot firing are as follows:—

1. No sparks are given off, and no naked lights are required.

2. Several shots can be fired absolutely simultaneously.

3. Shots can be fired from a distance, which is of great advantage in shaft-sinking, etc.

4. When a shot misses fire, the workmen can go straight back to work; whereas, if ordinary fuse has been used, a considerable delay is necessary.

5. The cartridge containing the detonator can be placed at the back of the hole, which is the most favourable position, as it tends to complete detonation of the charge.

On the other hand, the apparatus required for electric shot-firing is heavy and cumbrous. Electric fuses are more expensive than ordinary tape fuse; and when blasting in very hard stone, the cables are cut and damaged by almost every shot.

There are two systems of electric blasting in general use, viz. high tension and low tension.

In high-tension blasting a very small current at a high pressure is employed, and in low-tension blasting a comparatively large current at a low pressure is necessary. Neither of these systems has any very great advantage over the other. Low-tension are much more easily tested than high-tension fuses, and the cables for low-tension firing need not be so well insulated, and are therefore more efficient when worn or damaged than would be the case if the high-tension system were employed. The apparatus required for electric blasting are: Batteries, cables, electric detonators or fuses.

Batteries.—Electric exploders may be of two kinds:—(1) Primary, voltaic, or dry cells; (2) mechanical exploders.

Primary Cells consist of certain elements placed in jars containing an exciting liquid; usually metals and acids are employed, the electricity being generated by the chemical action of the acid upon the metal. These batteries are seldom, if ever, used for shot-firing, owing to their weight and cost, and to the inconvenience of carrying the liquid about.

Dry Cells.—These are cells which contain no liquids, and in which electricity is generated and given off when a circuit is made.

A well-known type is the Obach dry cell: it consists of a cylinder of zinc containing a central rod of carbon, together with depolarizing and exciting mixtures in the form of paste. These cells are made in various sizes. Size 0 is 6 inches by $4\frac{1}{2}$ inches by $4\frac{1}{2}$ inches, and costs about *2s. 6d.* Each cell gives off electricity at about $1\frac{1}{2}$ volts pressure, so that when two are coupled in series, the total pressure is 3 volts. Two or three are generally employed, being coupled in series and placed in a light box for convenience in carrying them about. As they are used, the discharge of electricity gradually decreases. One set will fire about 1000 shots before it becomes too weak and has to be discarded. As the voltage of these cells is low, they can, of course, be only employed in conjunction with low-tension detonators.

Mechanical Exploders.—These are small dynamos (Chapter XXIX.), worked by hand. The poles are usually permanent

magnets, and the armature is very rapidly rotated, by means of the handle, through gearing. The high-tension machines generate a very small quantity of electricity at a pressure of from 100 to 500 volts, and low-tension machines generate a larger quantity at a much smaller voltage.

Cables.—These consist of two insulated strands of copper wire, forming a single cable. A similar cable is used for high- or low-tension blasting.

Cables are subject to much rough usage, being frequently cut and damaged by the material dislodged by the shots. The insulation, too, gets worn away by their being dragged about from place to place. Broken strands can be roughly repaired by twisting the broken ends together, care being taken that the two strands cannot come together where both are bare.

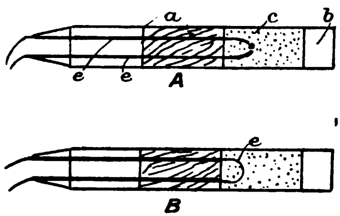


FIG. 64.—A, high-tension detonator; B, low-tension detonator.

Better insulation is required for high- than for low-tension blasting and for wet than for dry places.

Electric Fuses.—A, Fig. 64, shows a high-tension detonator. *a* is the copper cylinder containing the fulminate of mercury *b*, the priming of gunpowder *c*, and the insulated wires *e*, *e*.

These wires, which are surrounded by the priming where they terminate, do not quite touch each other, but are separated by a very small space, as shown. The electric current, which is generated by the exploder, passes from the exploder along one of the wires and back along the other.

As the ends of the wires do not touch, the current has to jump from one to the other, and pass through the priming, which offers high resistance. The interruption of the current thus caused results in a spark, which lights the priming and fires the fulminate.

Low-tension detonators (B, Fig. 64) are similar to those used for high-tension firing, except that the ends of the fuse

wires are connected by a "bridge" of very fine platinum wire, as shown at *e*. The platinum wire, being very thin, offers great resistance to the passage of the electric current, the result of which is that the wire is heated to redness, and so fires the priming. High-tension fuses may be compared with arc lamps—a small quantity of electricity is necessary, but it must be at a high pressure in order to leap from one wire to the other. Low-tension detonators may be compared with ordinary electric incandescent lamps, the wire being heated by the passage of a considerable quantity of electricity at a low pressure.

Low-tension detonators are tested by putting the fuse in a weak electric circuit. If the bridge is broken the current will not pass.

To make the test, the detonator should be slipped down a pipe through a hole in the wall of the room in which the tests are made; or it may be put through a small hole in an iron box, sufficiently strong to resist the force of an explosion, should the detonator explode by any mischance. The fuse wires are connected with a small battery and galvanometer. If the bridge is perfect the current passes through the fuse, and the needle of the galvanometer is deflected.

Firing the Shots.—The process of firing shots by electricity is as follows: The detonator is carefully pushed into the cartridge, and the wires are bent back and hitched round the cartridge, as shown at *a*, Fig. 65. This is not always done, but it is a good practice, as it prevents the detonators becoming withdrawn from the charge. The cartridge is next pushed gently into the hole, which has previously been cleaned by means of the scraper. The shot-firer then holds the wires in one hand and stems with the other, using the stemmer very lightly for the first few rounds. The wires are then connected with the cable in the manner shown at *b*, Fig. 65. The cable

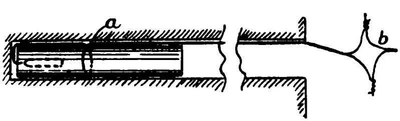


FIG. 65.—Charging shot-holes.

should be turned round a prop or secured with a weight a few yards from the hole, so that any pull on it will not affect the connection to the fuse. The cable is let out until a safe place is reached; the end is then connected to the exploder and the shot fired.

Misshots.—These may result from defective materials or from a short circuit. When a short circuit is formed, the current goes back to the exploder without passing through the detonator. When a shot misses, the battery should be disconnected from the cable, and the shot-firer should first carefully examine the connection between cable and fuse, to find out whether bare wires are touching each other at any point, or whether both bare wires are touching any conducting medium, such as water. If the junction between fuse and cable is good, the cable should be examined and raised out of any wet places there may be in the roadway. If a high tension battery is employed, it may be tested by making and breaking the contact between the terminals and a short piece of fuse wire. The cable can also be tested in the same way by uncoupling it from the fuse and sending a current through it. If these details are found to be in order, either the detonator is bad or the fuse wires may have become kinked in the hole and the insulation rubbed off.

When a good brand is used, the proportion of bad fuses is very small, being not more than one for every two or three thousand. When a miss-fire occurs, another hole must be drilled and fired, not nearer than 6 inches. The direction of all shot-holes should be indicated after boring, so that the second hole may not strike the first. The end of the fuse that has missed should be tied by a cord to a prop, so that it can be recovered after the coal or stone has been blown, otherwise it may be accidentally struck and exploded.

Simultaneous Blasting.—When several shots are to be fired simultaneously, they may be arranged in two ways. A, Fig. 66, shows the method of firing several shots arranged *in series* by means of a low-tension battery. The direction of the current is indicated by the arrows. It will be noticed that

the whole of the current goes through every shot in turn. The connection between the fuse wires is usually made with old fuse wires, which the shot-firers save for the purpose. When

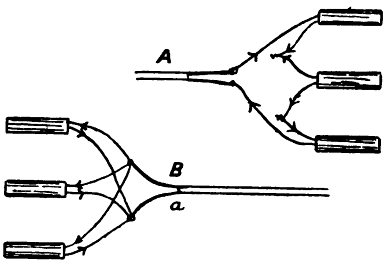


FIG. 66.—Simultaneous blasting: A, in series; B, in parallel.

the high-tension system of blasting is employed, the shots are arranged *in parallel*, as shown at B, Fig. 66. In this case the current splits at the end of the cable, a part of it only going through each shot.

CHAPTER XI.

METHODS OF WORK.

Choice of a Method of Work.—The two chief methods of working coal are by "*longwall*" and by "*pillar and stall.*" There are many modifications of each of these methods, some of which partake of the characteristics of both to such an extent that it is difficult to say to which class they belong. In the purest form of longwall, the whole of the coal is taken out at one operation, and no pillars are left, the roads to the face being maintained through the goaf. In the pillar-and-stall method of work, the coal is first cut up into pillars, which are subsequently extracted in "lifts;" that is, by narrow strips being worked off them. The longwall method is gradually displacing pillar and stall in almost every district, and in some important coal-fields it is exclusively employed.

In selecting the method by which a seam of coal is to be worked, there are many important factors to be taken into consideration.

The success of a colliery depends largely upon the underground costs, and upon the quality—as regards size—of the coal after it has been wrought, and both of these are greatly influenced by the method of working.

It often happens that the method of working suitable for one part of a mine is quite unsuited for the same seam in a different part, but, notwithstanding this, a similar method is employed throughout. The workings may become deeper, or the seam may alter in character or thickness, but as the change takes place very gradually, the original method may be followed until it has become quite unsuitable.

The choice of a method of work is governed by the following considerations:—

(*a*) *Locality of the Mine.*—The choice of a method of work is affected by the locality in which the seam is found. If the workmen and officials in the district are accustomed to any particular method, it would be unwise to make a new departure without good reasons, as it is very difficult to train men who have been accustomed to one method only to work efficiently in another. This difficulty is very serious, and can only be overcome by great patience, as men working under new conditions cannot at first get good results, and this leads to an increased cost of working the coal just at the time when the price-list has to be settled.

(*b*) *Markets.*—The purposes for which a coal is to be used have a very important bearing upon the manner in which it should be worked. If it is a house-coal and non-coking, the slack may be of little or no value, and every effort must be made to obtain as large a proportion of round coal as possible, which would be a very strong argument in favour of longwall, for with no other method of work is the coal got in such good condition. At some collieries, however, the whole output is ground up and coked, so that there is no difference in value between large coal and slack, in which case cheapness in working is the main consideration.

(*c*) *Thickness.*—Very thick seams are usually worked by some form of pillar and stall, though in some few cases longwall in two or three lifts is being practised, the lower portion being worked first and allowed to settle, and the upper portion being subsequently extracted. The thickness of coal taken out in a longwall face is limited in practice to about 7 feet; if the seam is thicker than this, the upper portion may be left to form a roof in the stall faces, and got down in the goaves when the back timber is drawn. Very thin seams are more often worked by longwall than by pillar and stall, though there are many exceptions. Seams between 2 and 3 feet in thickness are largely worked by pillar and stall in West Yorkshire and Lancashire:

(*d*) *Character of the Seam.*—When a seam is interstratified with dirt bands, it is better worked by longwall, as the dirt bands form packing material, and can be more easily disposed of than is the case in pillar-and-stall workings.

The disposal of the dirt is a very serious item in some seams, as very large quantities may have to be drawn to bank.

If the coal-field is very faulty, the system of pillar-and-stall working has certain advantages over longwall, for in longwall work no preliminary opening out or exploration is necessary, hence an unexpected fault may cut off a large proportion of the faces with very little warning, which could hardly be the case with the pillar-and-stall method.

(*e*) *Character of Roof and Floor.*—The nature of the roof and floor of coal-seams varies very greatly, and has the greatest possible influence upon the cost of working the coal, and upon the choice of the most suitable method of work.

A thick seam without dirt bands, and having a strong rock roof, is not adapted for longwall, as it makes no dirt for packing, and the cost of ripping down the hard roof for the gates is excessive. Moreover, the rippings will require blasting, and when a mine is gaseous this is very objectionable, and, owing to the Explosives Order, very inconvenient. When the coal-seam is hard and the floor soft, the pillar-and-stall method is unsuitable, as the weight of the superincumbent strata upon the hard coal presses down the floor under the pillars and squeezes it up wherever the coal has been extracted, causing the floor of the roads to "heave," and adding enormously to the cost of keeping open the roadways. Some seams are of such a nature as not to stand well in headings; sometimes the coal at either side grinds and crushes out, leaving the roads of abnormal width; or it may be almost impossible to keep up the roof owing to the grinding action breaking it up into small pieces for a great height. Seams of this character should of course be worked by longwall.

Inclination.—Seams lying at a high inclination may be worked either by longwall or by pillar and stall, the former

being the more common when the seams are thin, and the latter when they are thick.

Depth.—The depth at which a seam lies has a very important effect upon the system by which it should be worked. As the depth increases, the pressure upon the coal becomes greater, which causes the coal to be crushed and the roof to be difficult to keep up.

It is now generally recognized that seams lying at a great depth should be worked by longwall. There are still many collieries where deep seams are being wrought by pillar and stall, but most of these are old places in which the method was commenced when the seam was comparatively shallow, and has been continued without regard to the changed conditions.

The proportion of slack made in working a seam is generally greater when deep than when shallow; and as a rule the roads in deep mines are the more difficult to keep open. There are, however, exceptions to this, as the nature of the roof itself has to be considered.

The depth to some extent influences the choice of a method of work in another way. Owing to the high temperature of deep mines, it is of the utmost importance that the ventilation should be good, as by brisk ventilation only can the temperature of very deep mines be kept within working limits. The ventilation in longwall mines is much simpler and more thorough than in mines worked by pillar and stall; hence the temperature of the workings can be kept lower. The temperature of the strata is usually about 50 degrees Fahr. at a depth of about 50 feet, and the increase in temperature averages about 1 degree Fahr. for every 60 feet of increased depth. According to this rule, the temperature of the strata at 1000 feet in depth would be nearly 66 degrees Fahr.; at 2000 feet it would be 82½ degrees Fahr.; and at 3000 feet, 99 degrees Fahr. The temperature of the mine is usually several degrees lower than that of the strata, owing to the cooling action of the ventilation. The temperature in which men are able to work depends very greatly upon the dryness of the air. If the air is

humid as well as very hot, it is impossible for men to work, but if it is dry, very high temperatures can be withstood by men who have gradually become accustomed to the conditions. Sufficient data are not available to enable the limit of temperature to be definitely fixed, but it is probably about 100 degrees Fahr. The greatest depth below the surface that any English mine has reached is 3483 feet, which is the maximum depth of the Rams mine workings at Pendleton colliery. At that point the temperature of the strata is 100 degrees Fahr., and of the workings $92\frac{1}{2}$ degrees Fahr. At Agecroft colliery, at a depth of 2940 feet the temperature of the workings is 84 degrees, and of the strata 92 degrees.

It seems probable that the greatest depth at which coal can be economically mined with our present appliances is about 4000 feet.

The temperature is influenced by the nature of the overlying strata and the contour of the surface. There is also evidence which tends to show that the increase in temperature is not quite so rapid as the greater depths are reached.

Special Conditions.—There may be special difficulties to be overcome in the working of a seam which necessitate the adoption of a special method of work to meet them. For example, seams liable to spontaneous combustion should, if possible, be laid out and worked in such a manner as to reduce this liability and to limit the effects of gob fires should they occur. When other conditions are favourable, this may be done by some system of longwall retreating.

Sometimes seams have to be worked which are overlain by large volumes of water, either found on the surface, or contained in the strata, and a special method of working has to be adopted to prevent this water from making its way through the strata into the workings. In working under-sea coal, it is usual for the proportion of the seam which may be extracted to be settled by lease, and it has been found to be quite safe to extract the whole of the seam when the cover is about 100 yards thick. A strip of coal should be left alongside each fault, and in some cases, as an extra precaution, the

coal-field has been divided into panels by barriers of solid coal, so that if the sea-water should percolate into the mine, the district can be shut off by dams.

When coal is worked from under valuable buildings, less damage is occasioned by longwall than by pillar and stall. If the face advances regularly and continuously, the strata gradually subside without breaking, and no damage is done; but if the face stops, a break is formed, and great damage may be done to any buildings that happen to be on the line of this break. This is especially the case when the workings stop at an upthrow fault, in which case the line of break follows the slope of the fault.

Comparative Results of Longwall and Pillar and Stall.—The objects to be aimed at in selecting a method of work are—to get the coal safely, economically, in good condition for the market, and to get the whole of it. Discussing these in order—

Safety.—As regards safety, there is little to choose between the two methods; but what little advantage there is, is certainly with longwall. The ventilation in longwall pits is much the simpler, and is more efficient, as there are no "dead ends," whereas in pillar and stall a large proportion of the men are dependent upon some form of brattice for their supply of air. The risk of outbursts of gas is also much lessened by the longwall method, which is a very important matter in working certain seams.

Economy.—The comparative cost depends upon the suitability of the seam to the method by which it is worked. Some seams can be worked very cheaply by pillar and stall, and others quite as cheaply by longwall. In the former system the cost of heading and cutting has to be borne, as against the cost of ripping in the latter, and the relative cost of these items depends upon the characteristics of each individual seam.

Condition of the Coal when worked.—It is under this head that the longwall system compares most favourably with its rival, as there is no doubt that with ordinary conditions much

more round coal is obtained by longwall than by pillar and stall. In districts where the slack is of little or no value, this is of the utmost importance. The tendency is for slack to become of more value than formerly, so that it is probable that the great difference in value between large coal and slack will not be maintained. Coal which has stood for a long time in pillars is usually found to have deteriorated in quality, irrespective of size.

Getting the Whole of the Coal.—In seams of moderate thickness the whole of the coal should be got out, no matter what method of work is adopted; but in the pillar-and-stall method small portions of coal are frequently lost, whereas in the longwall system there may be absolutely no waste. When a seam is more than 6 or 7 feet in thickness there is nearly always a considerable amount of waste. Frequently the upper portion of the seam is left to form a roof in the face, and though it is supposed to be got from the goaf, a large percentage of it is unavoidably lost.

CHAPTER XII.

WORKING BY LONGWALL.

In the longwall method of working coal, the whole of the seam is extracted in a more or less straight face in one operation, the roof being allowed to sink as the workings advance. Access to the face is maintained by "gates," which are roads made through the "goaf," or old workings. Fig 67 shows the ordinary method of working coal by longwall. The road along

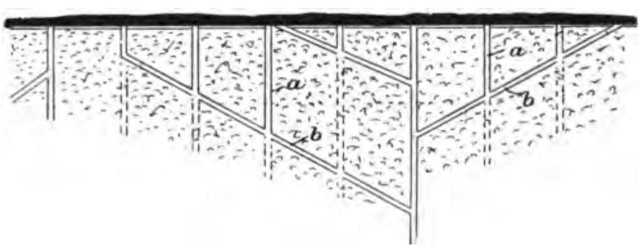


FIG. 67.—Longwall in seams of moderate inclination.

the face is kept open by being supported by two rows of timber.

If the roof is strong, props alone are set; if weak, bars are placed at right angles to the face from prop to prop; and if unusually heavy, wood chocks may be set alternately with ordinary props. The gates are kept open by means of "gate-end packs," which are built up of the material got from ripping down the roof in the gates. The gates require ripping, because the roof from under which the coal has been taken gradually

sinks and crushes up the packs. Intermediate packs are built between the gate-end packs to steady the roof down as it sinks, the width and distance apart of these intermediate packs depending upon the amount of material available with which to build them.

The operation of getting coal at a longwall face is conducted as follows :—

The coal is first "holed," or undercut, and is supported by sprags. It is then cut through or "broken into" opposite the gate, and several sprags are withdrawn; then, if the coal does not fall, it is wedged or blasted down and filled into corves, a corf road being laid along the face as soon as sufficient room is made for it by the removal of part of the "web" of holed coal. As the face is cleared, a fresh row of timber is set and the back row drawn, and intermediate packs built up. After a part of the web is cleared, holing is commenced at the cleared portion, so that in one part of the "stall," or "benk," the coal is being holed, and at another part it is being filled out.

The gate-roads are usually ripped and packed during the night-time.

To carry on the longwall system to perfection, each stall should be let to a set of men, who share their earnings and employ the holers and fillers. In some districts, where double shifts are worked, several men work independently in one stall, some working on one shift and some on the other. When this is the case, the proper holing and working of the stall is impossible, as the men on one shift naturally object to leaving coal holed for those who follow them to fill out and be paid for. In some seams the coal can be got without holing, and the practice of getting coal without systematically holing it appears to be on the increase, and has a detrimental effect upon the systematical carrying out of the longwall method of work.

Modifications of Longwall.—Seeing that the longwall method of working coal is employed under a great variety of conditions, it follows that it must have many modifications.

Direction of Face.—The direction of the face is mainly

governed by the cleat of the coal; it may also be influenced by the dip of the seam.

When the cleat lines are strongly marked, the coal generally gets most easily when worked on "bord," that is, when the face advances parallel to the cleat lines. On the other hand, more round coal is made when the face advances either "on end," that is, at right angles to the cleavage, or on "the cross," which is a direction between bord and end. For this reason, where the slack is of little value, the coal is usually worked on end or on the cross, and where it is not important to get a large proportion of round coal, the faces advance on bord.

Direction of the Gates and Roadways.—The roads in a long-wall pit consist of main engine planes, main gates, cross-gates, and ordinary gate roads. The gate roads are cut off at intervals by cross-gates, the old cross-gates by newer ones, and the main gates by the engine planes. This is necessary in order to keep down the length of the roads as much as possible. When the seam is flat, the direction of these roads is entirely a matter of convenience, but where the seam dips the question of gradient has to be considered. Figs. 68 and 69 show methods of working inclined seams, in one of which the face is level, and in the other it is carried at full dip.

Distance apart of Gates.—The distance apart of the gates depends upon the thickness of the seam, the nature of the roof, and the custom of the district. In flat seams the gates are carried in the centre of the face, but when the faces are inclined the rise side of the stall should be the longer, so that most of the coal comes downhill to the gate end. In thin seams the corves may have to be loaded at the gate end, as there may not be height enough for them in the face; this necessitates the coal being thrown back to the gate end, and when this has to be done the gates should not be more than 12 or 15 yards apart.

Where the coal is sufficiently thick to allow the corves to be taken into the face, the distance between the gates may be from 30 to 80 yards. If the gates are too far apart, the face advances very slowly, and unless the roof is good, falls

may occur; on the other hand, by bringing the gates too close together, the cost of ripping is increased. In the Midlands the average distance that longwall gates are apart is about 40 yards.

The distance apart of the cross-gates averages about 200 yards; it depends upon the manner in which the ordinary gates stand. When the latter becomes bad, a new cross-gate is set out to cut them off. Cross-gates are usually ripped and packed as the coal face advances, and are driven at an angle with the face. In some few cases it has been found preferable to scour the cross-gates through the goaf after it has settled.

In some methods of working inclined seams no cross-gates are required, the gates being cut off by the level above, as shown in Fig. 69.

Fig. 67 shows the ordinary method of working a comparatively flat seam by longwall. The example is taken from the Barnsley Bed, as worked in South Yorkshire, at depths varying between 300 and 700 yards, and having the following section:—

	ft.	in.	in.
Bags (coal)	1	0	
Top softs	2	0	
Clay seam dirt			2
Clay seam (poor)	0	8	
Hards (best steam coal)	2	6	
Bottom softs	2	0	
	8	2	

The whole of the "bags" and about half of the top softs are left up in the faces, and the gate-end packs are built under them. They are ripped down in the gates to make height for the traffic, and are filled out of the goaves between the packs, when the back timber is drawn.

When the roof above the coal is bad, it is liable to come down with the roof coal when the timber is drawn, causing it to be buried with dirt, and consequently lost. The yield per acre from a seam having the above section is about 10,000 tons, which is rather less than 1200 tons per foot thick per acre.

The gate roads (*a*, Fig. 67) are set out 33 yards apart, and are made 10 feet wide between the packs, which are about 12 feet wide. No intermediate packs are built, and all the back timber is drawn out, allowing the roof to break freely behind the second row of props. The cross-gates (*c*, Fig. 67) are put in when the ordinary gates begin to crush up; their distance apart varies, being from 100 to 200 yards.

In the deeper collieries it is found to be impossible to maintain a good road right up to the face, owing to the settlement of the strata, so that the main roads are not made to their full size at once, but are remade as the face gets about 100 yards in advance.

The thinner seams, when lying at a comparatively low inclination, are worked in a similar manner, except that the gates are usually rather further apart, intermediate packs are built, and, of course, no roof coal is left up in the faces. The rippings are usually made in two or three lifts; the first ripping is made close up to the face, and as the roof gets low a second ripping is started, and a third follows, if necessary. If the whole thickness of ripping were taken at once, the gates would be unnecessarily high towards the face when cut off by the cross-gates, but by taking the ripping at twice, only the older part of the gate has to be made the full height, as the end near the face is cut off by the cross-gate before it has had time to sink enough to necessitate a second ripping. In this system of working the opening out is automatic, as no roads are driven in advance, but all are made as the face advances. The only drawback to this is that in unproved ground an unexpected fault might cut off a large district with very little warning.

Longwall in Inclined Seams.—Fig. 68 shows a method of working by longwall when the inclination is considerable and the line of face is level. This example is taken from the Barnsley seam, in the neighbourhood of Sheffield, where it has a section of about 4 feet 6 inches of good coal. A main engine plane is driven to the full dip of the seam, and from it levels are set out on either side. These levels are about 200 yards

apart; the gate roads are 60 yards apart, and advance to the rise from the levels. The whole of the coal is taken out in the levels as shown in the figure, and about 20 yards are taken out below them, so as to allow the roof to settle down gradually without breaking. No cross-gates are employed, and the faces are "stepped" as shown; that is, each stall is about 10 yards behind the one it follows.

The object of stepping faces is to localize the weight, so that if the roof breaks away in one place, the damage may be confined to the one stall. It is probable that these "steps" are a mistake, and the tendency is to do away with them. They increase the proportion of small coal, and interfere with

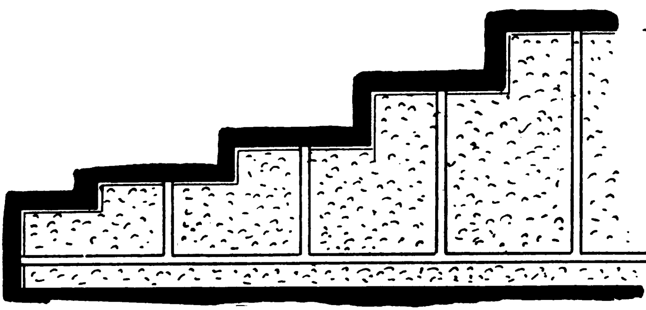


FIG. 68.—Longwall in inclined seams, faces level.

the proper regulation of the weight upon the face; moreover, it is very difficult to keep the cuttings anything like the proper length. If the men in one stall habitually send out more coal than their neighbours, as frequently happens, their stall gets a long way in advance of the others, and the cuttings gradually become very long, and are liable to break down and obstruct the ventilation. The gate roads are made 10 feet wide, and the ripping is taken 5 feet thick, the full thickness being ripped at one lift. The haulage down the gates is entirely by self-acting inclines, so that it is expensive to take a second ripping, because the stuff that is ripped down cannot conveniently be sent uphill into the stalls, but has to be sent

downhill to the levels, and may have to be sent out of the pit.

In most collieries a really good gob road cannot be made until practically the whole of the road is in the roof. That is to say, if a road 6 feet high is required, it will not be satisfactory until about 6 feet of ripping has been taken down, and in some cases a good deal more may be necessary.

When the inclination is great, and the most favourable line for the faces, as regards cleavage, is steep, the method of work shown in Fig. 69 may be adopted. This example is taken from

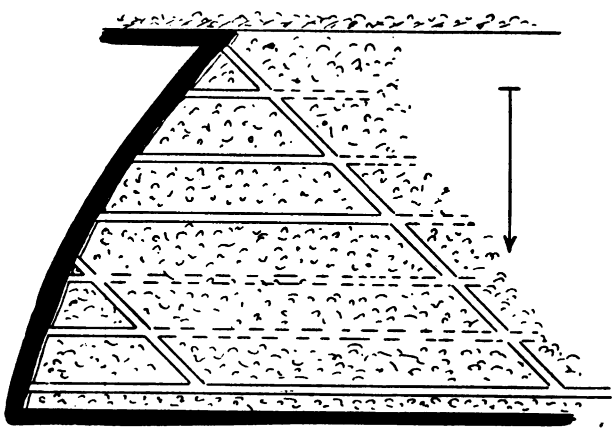


FIG. 69.—Longwall in inclined seams, gate roads level.

workings in the Arley Mine in West Lancashire, the thickness of the seam being 3 feet, and the depth from the surface about 400 yards. The main levels are 100 yards apart, and from them cross-gates are set out at an angle of about 45 degrees. From these cross-gates the ordinary gate roads are set out parallel to the main level. The gates are 15 yards apart, and each stall has 3 yards of face to the dip and 12 yards to the rise. The corves are not taken into the faces, but are filled at the gate ends, the coal being cast back to them. When full, they are trammed to the cross-gate and lowered to the main level,

either by self-acting inclines or by balance jigs. Very little ripping is taken in the gates, as they are cut off by a new cross-gate when they have advanced 60 yards, and as the stalls are so short the advance is very rapid. This method is not suited to a tender coal, as much slack is made by casting back the coal to the gates. In the example given above, about 50 per cent. of the output is slack, although the seam is of a fairly hard nature.

A modification of this method is illustrated by Fig. 70, which is taken from workings in the Silkstone seam (South Yorkshire), where it has a thickness of 3 feet 3 inches, and lies

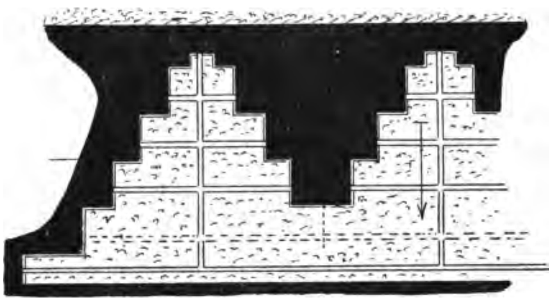


FIG. 70.—Longwall in inclined seams, gates level.

at an inclination of 1 in 5. In this modification the main levels are driven out in advance, and not opened out in the course of the working, as in the last example. Every 200 yards along these levels "jinneys" are set out to the full rise of the seam, and from either side of these "jinneys" level gates are set off. The gates are 20 yards apart, and each stall has 5 yards of dip and 15 yards of rise coal. Each stall goes 100 yards before it is finished by meeting the stall set off from the next "jinney"; and as one pair of stalls finishes, another pair is started from the opening-out "jinney" as shown. This method requires a large amount of pit room, as it is not convenient to have more than three or four pairs of gates from one "jinney." The working of self-acting inclines or "jinneys" is fully described in Chapter XXIV.

Longwall with Gates in the Solid.—When the roof of a coal-seam consists of a thick bed of hard rock, special difficulties present themselves in working the seam by longwall. All the rippings have to be blasted, and no material can be obtained with which to build the intermediate packs. It is also very difficult to regulate the weight upon the coal face, as the roof will not break when the timber is withdrawn, but stands for a long way back in the goaves, and comes down over large areas when it does fall, breaking all the timber and closing the stalls and gates. The method shown in Fig. 71 has been adopted to reduce the difficulty of making and maintaining the gateways in a seam having a roof of this description. This example is taken from a seam which has the following section, the depth from the surface being about 300 yards:—

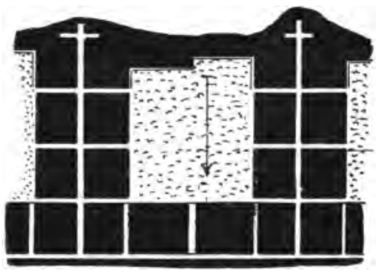


FIG. 71.—Longwall with gates in solid coal.

	ft.	in.	in.
Inferior coal	0	6	
Dirt			4
Brights	1	3	
Hards	2	1	
Bottom brights	1	2	
	5	0	

The roof consists of beds of strong rock having an aggregate thickness of about 30 yards.

The main levels consist of headings in the solid coal, and are driven about 150 yards apart. Rise headings are started off these main levels; they are about 100 yards apart, and serve as gate roads. A pillar of coal 22 yards wide is left on either side of these rise headings, and the coal between these

pillars, which is about 55 yards wide, is worked uphill in a longwall face. The face and headings are driven up simultaneously, the headings being kept a little in advance, and slits put through for the passage of the coal. These slits are about 25 yards apart, and the coal from the face is brought down the cutting side and along the top slit into the gate. The gate road pillars are extracted after the stalls are finished.

In this method of work no ripping is necessary, and the gate roads are protected from the weight by means of the pillars on either side. These pillars also serve to prevent the effects of a "weight" upon one stall spreading to the others. The cost of the headings is considerable, but against it must be set the cost of the ripping and packing, which would be required if the gates were taken through the goaf, as is done in ordinary longwall. This modification of longwall is also applicable to seams having an unusually weak roof. It was adopted many years ago in working the Wathwood coal in South Yorkshire, where it was about 4 feet 6 inches in thickness, and lay very near to the surface. The roof was found to be so wet and weak that it was almost impossible to keep the gates open when all the coal was taken out, and they were carried on packs in the usual way. This difficulty was overcome by making the gates in the solid, as shown in Fig. 71.

Longwall Retreating.—In this method, headings are driven to the boundary of the district, and the coal worked back by a longwall face, leaving the goaf behind. Longwall retreating is specially suitable for working seams which are liable to spontaneous combustion, as the goaf is left behind, and is gradually compressed almost solid by the weight of the strata settling upon it. A district must be headed out before any large output can be obtained, and this is necessarily a work requiring considerable time. This is one of the objections to this method; but it is not necessary to head out right to the boundary of the royalty, but only to the boundary of a district, the extent of which can be arranged to suit circumstances.

It is only in certain seams that this system of work is successful, or even possible; in some of the deeper seams it is

almost impossible to drive headings, as they crush up as soon as driven.

Fig. 72 shows the method of working the main coal in South Derbyshire. A section of this seam is given on p. 37. The workable portion is about 7 feet in thickness, and is that portion between the "grounds" and "overcoal." This seam is extremely liable to spontaneous combustion. Formerly it was worked by the ordinary system of longwall, and attempts were made to isolate the goaves by building "wax walls" on each side of the gate roads. These "wax walls" were continuous walls, built up of plastic clay; they were compressed by the weight into solid barriers of hard

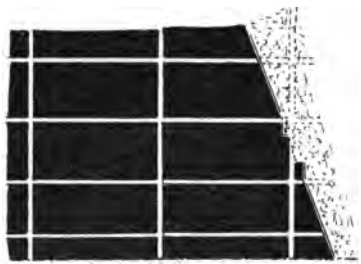


FIG. 72.—Longwall retreating.

clay, but only partially succeeded in keeping the air from the goaves, and the method of work was altered to that shown in Fig. 72.

Parallel headings are driven 40 yards apart to serve as gate roads, and slits are driven at intervals to connect them for ventilation.

The slits should be as few in number as possible, as they are expensive to cross, and cut up the faces. After the headings have reached the boundary, they are connected by a cross-heading and the coal worked back, leaving the goaf behind. The line of face should make an angle with the slits, so that they may be crossed gradually. The roof of the seam is the "overcoal," which is of inferior quality, and is usually left in

the pit; all the timber is drawn from the goaves, and packs of coal are built at the face to steady the roof down. As the face advances, a considerable amount of weight is thrown forward on to the gate ends, and they have to be very well timbered to prevent their being crushed up.

Longwall retreating has been adopted in a few cases in thin seams worked by coal-cutting machines, but, except under special conditions, it has not been largely employed.

CHAPTER XIII.

METHODS OF WORKING BY PILLAR AND STALL.

THE method of working coal by pillar and stall was introduced in the earliest days of coal-mining, when the pits were shallow and the workings on a limited scale. The method, in its simplest form, consists of cutting the coal up into rectangular blocks or pillars, which are afterwards worked out in lifts. When the method was first practised all the pillars were left unworked, and the object was to get as much of the coal as possible without bringing down the roof. With this object in view, the pillars were made very small, and the roads between them very wide. The result of leaving these small pillars was that as more and more coal was got out they were crushed up by the weight, and what was known as the "creep" was brought on. "Creep," then, results from the pillars not being left of sufficient size and strength to support the overlying strata; they are consequently ground up, and the roads are set "on the move," breaking the timber and bringing down the roof. In the old collieries the main road pillars were left of similar size to the others, so that there was nothing to prevent the "creep" from extending throughout the whole of the pit, and this frequently happened, resulting in the loss of the mine. The first improvement consisted of dividing the workings into panels, separated from each other by strong pillars of coal; the pillars confined the creep to one panel. The next great improvement was to lay the workings out in such a manner as to enable the whole of the coal to be extracted. This was

done by making the pillars much larger, and only taking out a small proportion of the coal in the first working, the pillars being afterwards got out in lifts.

There are many variations of pillar-and-stall working, some of the most important of which are described in the following pages.

Pillar and Stall as practised in the Durham Coal-field.—The first working, or "working in the whole," consists of cutting up the coal into rectangular blocks or pillars by means of roads driven at right angles to each other. These roads are termed "walls, or endings," and "bords," the endings being roads driven parallel to the cleat, and the bords roads driven at right angles.

In some cases the bords are driven wider than the endings, and in others both are driven of the same width. The second working, or "working in the broken," consists of getting out the pillars formed by the first working.

In some collieries a large area is cut up into pillars, which may be left for many years before being extracted; at other places the pillars are worked out soon after they have been formed, and before the bords and endings have had time to close up. This is known as "following up the whole with the broken." When pillars are left for a long time before being extracted the quality of the coal deteriorates, and the roads between the pillars become entirely closed up. These drawbacks have led to the practice of following up the whole with the broken becoming more common.

Fig. 73 shows the method adopted for working out the pillars at a Durham colliery. The seam worked is the main coal, which lies at a depth of about 300 yards, and has the following section, post roof :—

	ft.	in.	in.
Top coal	0	9	
Band			1
Bottom coal	3	9	
Splint coal (not worked)...			8
Workable section	4	6	

The bords A and walls B are driven 4 yards wide, and form pillars of coal 36 yards square, the proportion of coal taken out in the first working being about 19 per cent. of the whole. As the bords and endings have fallen in before the pillars are attacked skirtings, *s*, *s*, are driven along two sides of the pillar, which is to be extracted to open it out. A "siding over," *k*, is then driven as shown, to shorten the length of the juds. The pillar is then worked out by lifts, or "juds," 6 yards wide, as shown at J[1], J[2], etc. The last slice of coal is taken out in short lifts driven at right angles to its length.

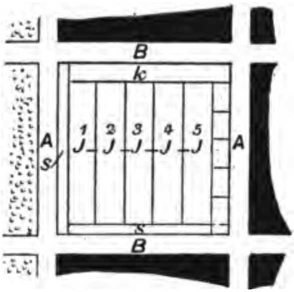


FIG. 73.—Pillar and stall.

There are many other methods of working pillars; in some cases the pillar is split, and juds taken off in both directions. The size of the pillars varies considerably; they should be made larger as the depth increases, because the weight upon them, due to the pressure of the superincumbent strata, also becomes greater.

A road is kept open to the face of the juds alongside the solid coal by means of props and bars. After a jud is finished, the road is pulled up and all the timber withdrawn, allowing the roof to fall. In taking out a large number of pillars, the line of goaf should not be straight, but zigzag, so as to break the weight; but care should be taken that no pillar lags behind, or it may become crushed and lost.

The Barnsley Bank System.—Fig. 74 shows the method by which the Barnsley Bed has been largely worked, where found at a moderate depth and inclination. Levels are driven in sets of three, leaving about 120 yards of coal between each set. The three levels which form a set are each 20 yards apart; the top level is for the purpose of opening the banks off, the middle one is for haulage, and the third and lowest is used for ventilation, and for opening off the return banks. From these levels,

banks 20 yards wide are set off uphill, at intervals of 60 yards. These banks are taken up to the lowest level of the set above, and when they reach this another 20-yard bank is opened off from it, and brought downhill as shown.

This system of work was very successful where the coal was shallow, and there was little weight. It is now, however, almost entirely superseded by the long-wall system of work,

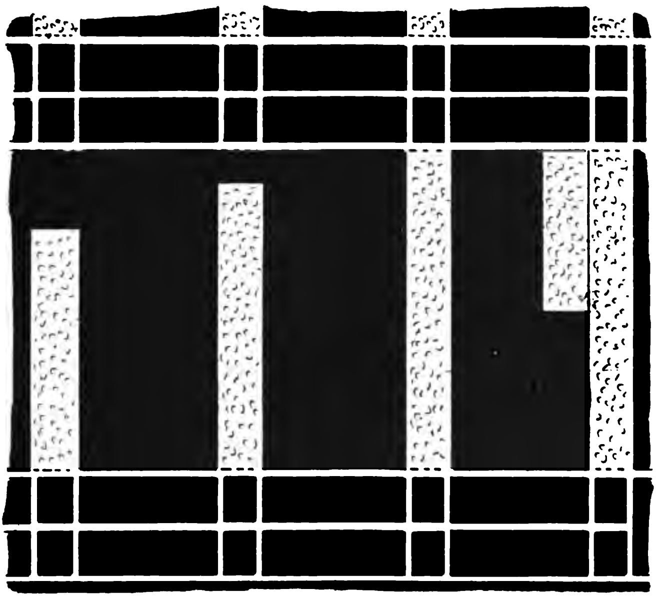


FIG. 74.—Barnsley Bank method of work.

which is found to be much superior where the depth is considerable, and there is now very little Barnsley coal remaining unworked at a depth of less than 400 yards.

Bord-and-Pillar Method.—Fig. 75 shows an important method of bord-and-pillar working.

The example is taken from workings in the West Yorkshire Beeston Bed, where it has a thickness of about 4 feet 9 inches, and lies at a depth of about 200 yards. Endings about 8 feet wide

are driven from the main level at intervals of about 40 yards. Bords are driven from ending to ending; they are set off narrow, but widen out to 5 yards within a few feet from the endings. The bords and endings are driven simultaneously, and the pillars between the bords, which are 40 yards square, are worked out in 5-yard lifts soon after they are formed, and before the endings have fallen in. The object of starting the bords narrow is to avoid the damage that would be caused to the endings by setting them off at their full width, and the reason that they are widened out is to decrease the cost of the "yardage" which has to be paid for driving narrow roads.

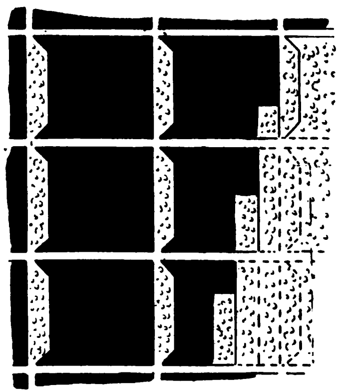


FIG. 75.—Bord-and-pillar work.

Some of the thin seams which occur in the lower coal-measures are worked in West Yorkshire and Lancashire by a method similar to the one illustrated in Fig. 75, the chief difference being that both bords and endings are driven rather nearer together. The corves used carry up to about 5 cwts. of coal, and stand about 2 feet high. In seams 30 inches in thickness, a few inches only of roof are ripped down in the bords and endings, and the corves are pushed or "trammed" from the face to the haulage roads. The men and officials travel about the pits on small flat-bottomed trams, as the roads are too low to admit of walking. This method, when applied to thin seams, results in a saving in the cost of ripping and packing, but the cost of tramming is very heavy, and much small coal is made.

The Double-stall Method of Work.—Fig. 76 shows the double-stall method of working coal. This method has been adopted to a limited extent in several of our coal-fields. Its

chief advantage over the single-stall method is that ventilation is made easier and more complete.

The example is taken from the workings in a seam 6½ feet thick, lying at a depth of about 300 yards. The levels are driven in pairs, and from these levels headings are taken up at the full rise of the seam in sets of three. These sets of rise headings are separated by about 250 yards of coal, and the headings in each set are 20 yards apart. The centre heading of a set is reserved for haulage, and the stalls are opened out from the outer ones. The headings are connected every 20

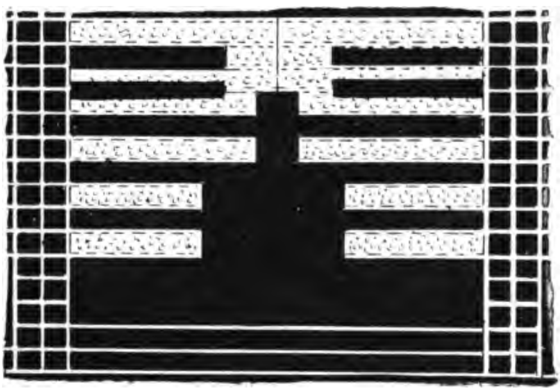


FIG. 76.—Double-stall method of work.

yards by slits, along which the coal is conveyed to the haulage road from the stalls.

The stalls are 20 yards wide, and leave a pillar of coal 20 yards wide between them. They are driven up in pairs, and when they have reached their limit—which is half the distance between the sets of headings—the 20-yard pillar is cut through and worked back. A road is made at each side of the stalls, and is kept open by timber and packs, but the timber is drawn from the space between the roads, and the roof allowed to fall.

After a district is finished, the coal along either side of the haulage road is worked out to the level.

Working Coal in Lifts.—Fig. 77 shows a method that is sometimes adopted for working steep seams. Levels are driven about 12 to 15 yards apart, and the coal between these levels is worked out in lifts 3 or 4 yards wide.

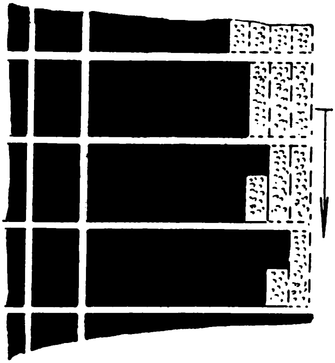


FIG. 77.—Working coal in lifts.

The workings in each level are kept a lift or two in advance of the workings in the level below, in order to keep the road intact.

The corves are not taken to the face, but are filled as they stand in the levels.

When thin seams are worked in this manner, the levels are driven wide, and a road ripped on their rise side, the dirt made from the rippings being packed at the low side of the level.

Beds of fireclay are frequently worked in this manner, and under ordinary conditions the cost is low.

Calculating Proportions of Whole and Broken Coal.—The percentage of coal obtained from the whole and broken workings may be calculated as follows:—

A mine has pillars 40 yards square, with bords 5 yards and endings 3 yards in width: what is the percentage gotten in the first working?

It is only necessary to consider the case of one pillar and one bord and ending, as the whole mine may be divided into similar figures.

Then original area of figure, $45 \times 43 = 1935$ sq. yards

Area of pillar, $40 \times 40 = 1600$

Area taken in first working $= 335$

Percentage taken in first working, $\frac{335 \times 100}{1935} = 17 \cdot 3$ per cent.

CHAPTER XIV.

SPECIAL METHODS OF WORK.

THE methods of work described in the previous chapters are only suitable for working seams which occur under normal conditions as to thickness, inclination, etc. But in various parts of this and other countries, seams are found to occur under such abnormal conditions as to render the adoption of a special method of work absolutely necessary.

Methods of working Steep Seams.—Seams which lie at an angle of about 45 degrees, or 1 in 1, can be worked by some modification of the methods already described; but when this rate of inclination is greatly exceeded, a special method of work has to be adopted. These very steep seams are termed "rearers," and are found chiefly in North Staffordshire and locally in Somersetshire and parts of South Wales and Scotland. Steep measures are usually found at the edges of basins, and when followed down the inclination gradually decreases. There are collieries in North Staffordshire working seams which are both vertical and horizontal in a comparatively small area. When seams are nearly vertical, the method by which they are worked is somewhat similar to the methods adopted for working mineral veins, which are usually nearly vertical.

The cost of working steep seams is not necessarily much greater than the cost of working the same seams when flat, but unless the seam is of a very strong nature, much small coal is made.

In Fig. 78, A shows a vertical and B a transverse section

of the workings of a thin seam of coal lying at a high inclination. The seam from which the example is taken is about 3 feet in thickness, and lies at an inclination of nearly 70 degrees from the horizontal. It should be remembered that A, Fig. 78, is not a plan of the workings, but a vertical section, and in dealing with excessively steep seams, a much better idea as to the character of the working is obtained from a vertical section than from a horizontal plan.

Levels are driven, 60 yards apart, from which the coal is

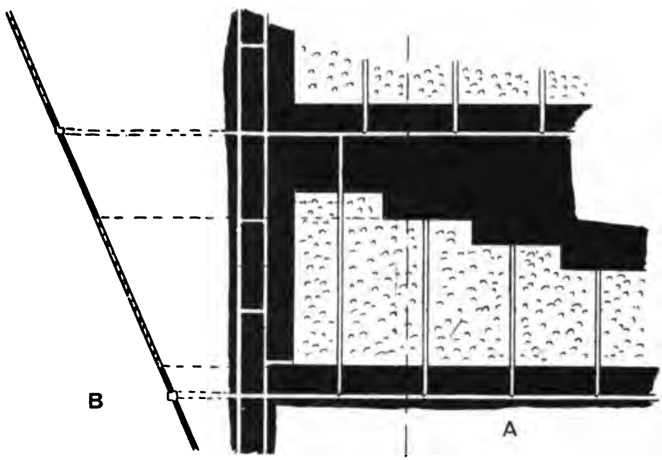


FIG. 78.—Method of working a thin seam lying at a high inclination.

worked to the rise, a pillar 20 feet thick being left above each level to keep back the goaf. As the seam is so nearly vertical, the coal-seam itself forms the roof and floor of the levels, as shown in the transverse section. Gate roads are set off every 20 yards along the levels, and are ripped and packed in the usual manner. These gate roads are divided longitudinally by a timber brattice, one side forming a travelling road to enable the men to pass from the levels to the faces, and the other forming a shoot for the coal. The shoot terminates in a wooden hopper, which is opened and closed at will by an iron

slide, so that the corves can be filled by pushing them under the hopper and pulling out the slide.

The shoot is always kept full of coal, in order to reduce the great breakage that would ensue if lumps of coal were dropped from the top to the bottom of the shoot. The men at the coal-face work on a platform built up of boards placed across timbers let into the strata. These platforms also serve to prevent the coal from falling into the goaves and being lost.

In some places where a similar method is employed, no pillars are left above the levels. When this is the case, the levels must be very carefully timbered, because the goaf, which forms the roof of the levels, may be very small and loose, and, if it should break through into the levels, may be difficult to secure.

The thick seams in Pennsylvania, when lying at a high inclination, are sometimes worked by a method of pillar and stall. Breasts 10 or 12 yards wide are taken up from the levels, each breast being separated from the next by a pillar of coal 10 or 12 yards thick.

A small space on either side of each breast is timbered off for a travelling road, and the remaining portion is kept full of loose coal. This loose coal forms a floor for the men to work upon, and avoids the use of scaffolding. After a breast is finished, the loose coal is drawn off and the roof allowed to fall in.

North Staffordshire Method of working Rearer Coals.—A very interesting method of working steep seams is practised in North Staffordshire. The seams are won by "cruts," or level drifts, which are driven from the shafts, as shown in Fig. 56.

After the coal has been reached, levels are driven out in it to right and left in pairs, the upper being used for haulage, and the lower for a return airway and for a water-level. From these levels rise headings are started in pairs (Fig. 79), each pair being separated from the next by 200 yards of coal. The distance between the main levels is about 120 yards, and all the coal is worked to the rise of the upper levels before any of the pillars are removed from the levels below. The rise

headings are carried up to within 10 yards of the goaf above, which forms the boundary of the breadth of coal which is to be worked.

From these rise headings, three levels are driven on either side for a distance of 100 yards; the levels are 10 yards apart, and 10 yards of coal is left between the top level and the goaf. The coal is then worked in lifts uphill from level to level. Operations are commenced at the top level first, by taking a lift almost up to the goaf above. The thin barrier of coal which is left next to the goaf is blown out by a shot, and the goaf comes sliding down into the lift. This is done to

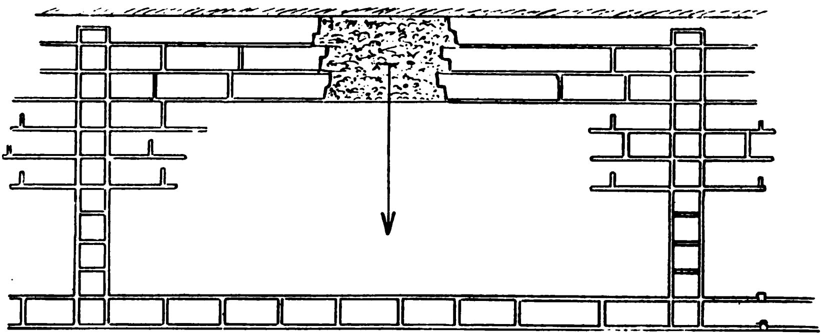


FIG. 79.—Method of working rearer coal-seams in North Staffordshire.

give the men something to stand upon whilst at work. The same method is pursued in the two levels below, the top levels being kept a little in advance, as shown in Fig. 79, which is a vertical section through a district. The coal is filled into corves, which stand in the levels and are not taken to the face. This method is somewhat similar to that illustrated in Fig. 77, except that in the case of these very steep seams the goaf is brought down from level to level, to give the workmen something to stand upon. The corves are raised and lowered in the headings by means of cage dips. Four lines of rails are laid in the headings, and a wheel fitted with a powerful brake is set up at the top. The cage, which runs on wheels and

carries one corf only, is attached to one end of a rope which passes round a wheel and has a balance weight secured to its other end. The cage and full corf are heavier than the balance weight, so that when the cage is loaded with a full corf, it runs to the bottom and pulls the balance weight up.

When the full corf is run off the cage and an empty one substituted for it, the balance weight is the heavier, so that it runs down to the level and pulls the cage and empty corf up.

The movement is controlled by the brake on the wheel, and the cage can be stopped at any of the levels. Whilst the pillars are being worked back in the manner described, fresh levels are being driven out below, as shown in Fig. 79. When all the seams in a breadth are becoming worked out, the shafts are sunk another 120 yards, and a lower breadth opened out by a drift.

Methods of working Very Thick Seams.—Excessively thick seams are always difficult and dangerous to work, and when, as is very often the case, they are liable to spontaneous combustion (see Chapter XXVIII.), both the difficulty and the danger are greatly increased. The thickest seam worked in England is the Ten Yard Coal of South Staffordshire, which has a maximum thickness of about 33 feet of clean coal.

Thick seams are also found in other districts, but in most cases a part only of the seam is workable, the remainder being left in the pit.

The working of a thick seam is usually attended by great waste. In Staffordshire, it is the custom to work the Ten Yard Coal two or even three times over. An area is worked and abandoned, and years after the same area is worked over again by systematically exploring the goaves, and extracting the pillars, ribs, and roof coal which were left behind in the first working.

There are two methods by which thick seams may be worked: (1) By square work, in which the whole thickness of the coal is worked in one operation; (2) longwall work in

several lifts, in which the seam is removed in several thicknesses, layer by layer.

The South Stafford Method of Square Work.—By this method the coal is worked in rectangular chambers or "sides of work," and as the Ten Yard Coal is extremely liable to spontaneous combustion, each side of work is enclosed by a barrier of coal 10 yards thick.

The inside dimensions of a side of work are about 46 yards by 64 yards. The roof is supported by six pillars of coal, each being 8 yards square, and surrounded on all sides by 10 yards of goaf. Fig. 80 shows a side of work when finished; it would be opened out and worked as follows:—

Narrow gate roads, *a*, *a*, are first driven in the bottom part of the seam, andare connected by the cross-road *b*. As soon as the connection is made, the coal from *b* to the boundary is worked out by longwall, making *b* 10 yards wide. This also is done in the bottom 6 or 8 feet of the seam, as is all the opening out. Whilst this is being

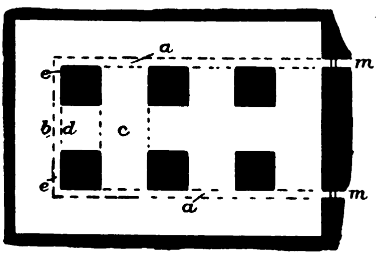


FIG. 80.—South Staffordshire square work.

done, the piece of coal marked *c* is taken out, and the gate roads *a*, *a* are also widened out. The piece marked *d* is next removed, leaving the two pillars *e*, *e* 8 yards square, 10 yards apart, and 10 yards from the boundaries of the side of work. Whilst this is going on in the bottom layer of coal, the upper beds are being worked at *b* by cutting through and dropping the layers one by one. The remainder of the pillars are formed in a similar manner, and the coal surrounding them dropped and filled out until all has been worked except the pillars; dams are then put in at *m*, *m*, and the side of work abandoned. Sometimes the pillars are got out by building large wood chocks between them to carry the roof.

In practice, this method is modified very greatly to suit

circumstances. Often the sides of work catch fire, and have to be closed before all the coal is worked.

The proportion of coal which is lost depends upon the nature of the roof, the depth of the seam, and the skill and care which is exercised. As a rule, only about 50 per cent. of the coal is got out in the first working.

Working Thick Seams by Longwall in Lifts.—A thick seam of coal is always made up of distinct bands or beds, so that it may be regarded as a series of thin seams lying close together.

The bands which form a thick seam may be worked separately by longwall, either in descending order by working the top band first, or in ascending order by commencing with the bottom bed. When seams are worked by lifts in this manner, one band may be worked over a large area, and the strata allowed to settle before attacking the next, or the face in one band may be closely followed by the workings in another. In either case the method is very similar to the methods adopted for working several contiguous or nearly contiguous coal-seams.

Methods of working Seams lying close together.—When two seams lie close together, or are separated by a thin band of dirt, the lower seam may be worked by the ordinary longwall method, and the upper seam filled out of the goaf by being dropped between the packs when the timber is drawn. This method results in much waste, as the upper seam is left over all the packs, and, if the roof above it is not very strong, it falls with the coal and buries it.

When the seams are separated by a dirt band of several feet in thickness, the lower seam may be worked first by longwall, the dirt between the seams being ripped down in the gates. After the workings in the lower seam have reached their limit, the upper seam is opened out and worked back, using the same gates as were employed to work the lower bed. This method is illustrated in Fig. 81, which shows the manner by which a seam is worked where locally divided by a dirt band, and has the following section :—

		ft.	in.
Top coal	... ...	2	8
Dirt	... ... from 3 feet to	6	0
Bottom coal	... ...	2	10

The gate roads are driven 20 yards apart, and are cut off when they have proceeded 100 yards. The first working is in the lower seam, the dirt band forming the roof in the faces, but being ripped down in the gates.

After a stall in the lower seam has reached the boundary,

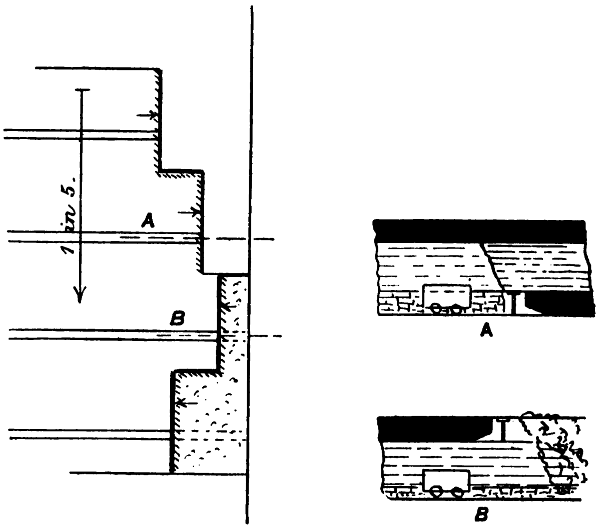


FIG. 81.—Plan and sections of method of working seams lying close together.

the top coal is cut through in the gate, a face opened out by a heading and worked back by longwall, the coal being thrown back into corves which stand in the gate.

A, Fig. 81, is a section through the gate and face in the first working, and shows the packs, ripping, upper coal-roof, and face in the lower bed. B, Fig. 81, is a section through the gate and face as it appears when the top coal is being worked back.

Under suitable conditions, this method works very well. The upper seam makes good coal, and is little damaged by the working of the lower bed.

Warwickshire Method of working Contiguous Coal-seams.—In the southern part of the Warwickshire coal-field, several seams are found lying very close together, being separated from each other by only a few feet of dirt. As the coal is very liable to spontaneous combustion, headings are driven down to the dip, and the coal worked backwards uphill, the goaf being allowed to fill with water. In some of the collieries, the Two Yard, Rider, Ell, and Slate coals are all worked together; in other collieries only two or three of the seams are worked. The method of working is as follows:—

Roads are driven to the full dip of the measures in the lowest of the seams which is to be worked. These dip roads are about 150 yards apart, and are driven for a distance of 500 or 600 yards. After the dip headings have reached their boundary, a "congate," or level cross-measures drift, is driven to intersect the whole of the seams. Levels are then driven in each seam from either side of the congate, to open out stalls in each seam, all of which are worked simultaneously. The faces in each seam are kept 15 or 20 yards apart, and the gate ends are allowed to hang back as far as possible, as when they advance a new congate is required.

CHAPTER XV.

TIMBERING.

Supporting the Roof and Sides.—In those parts of a mine which have to be kept open for transit or for ventilation, the roof, when weak, must be supported by timber or some substitute. The supports in the roadways are of a permanent character, but the supports in the faces are only temporary, and are withdrawn as the face advances.

The best timbers for mining purposes are Baltic, Norwegian, and Scotch pines, firs, and larches.

Timber should be dry when set, or a considerable portion of its strength is lost; it is usually found to be more durable when set without the bark. The cost of mining timber varies from about 8*d.* up to about 1*s.* 6*d.* per cubic foot; and the cost on the output varies in different mines between about 1*d.* and 8*d.* per ton. This great difference in cost is due to the nature of the roof and floor, thickness of seam, and method of working.

Preserving Timber.—In some mines the timber rots very quickly. It may be preserved by forcing creosote into the pores. Creosote is a mixture of pitch, creosote oil, naphtha, and ammonia; the timber is thoroughly dried, and the creosote is heated and forced into it at a high pressure.

One drawback to this method of treating timber is, that it causes the timber to be highly inflammable.

Another method of preserving pit timber is that known as Aitkin's process, in which the timber, after being thoroughly dried, is boiled for two days in a solution containing 7 parts of

common salt and 1 part of chloride of lime. Timber treated by this process will not burn, but is slightly weakened, owing to the presence of moisture, and is also considerably increased in weight.

The weight of pit timber varies between 40 and 50 lbs. per cubic foot.

Object of Timbering.—The object of timbering is not only to keep a roof up when it is bad, but to prevent it from becoming bad when it is sound. It is often cheaper in the end to pull loose stuff down rather than keep it up with timber, though, of course, there is a limit to this. Many roofs break in the form of a natural arch, after which they stand indefinitely with little timber. When this is the case, the best roads are obtained by allowing them to arch themselves over in this manner. Timber in the working places has now to be set at stated intervals, these being fixed at each colliery by special rules. In some districts it is the custom for the coal-getters to set and draw the timber in their working places; but in other districts this is done by men specially employed for the purpose.

Props.—Props are exposed to a *compressive* strain; their length is usually about ten or twelve times their diameter. Thus a prop 5 feet in length would have a diameter of about 5 or 6 inches.

The strength of a prop having these proportions depends upon its sectional area, and is about $1\frac{1}{2}$ ton per square inch. Thus a prop 6 inches in diameter has an area of $6 \times 6 \times 0{\cdot}7854 = 28{\cdot}27$ square inches, and a breaking strain of $28{\cdot}27 \times 1\frac{1}{2} =$ about 42 tons. If the prop is much longer than 10 diameters, its strength is decreased, as it breaks by being buckled and not by being crushed. To get the maximum strength, a prop should have flat ends, cut off square, upon which the pressure should bear evenly; it should be set at right angles to the pressure, and should have a cap or lid of soft wood between it and the roof. In inclined seams props should be set almost at right angles to the roof and floor; the tops should lean uphill a little from this line, because, when

the weight comes on, it tends to force the tops of the props downhill; and if they were set perfectly square in the first instance, this might push them out, whereas by "undersetting" them a little, the effect of the pressure acting downhill is to tighten them. To set a prop, the workman "stamps" a depression into the floor with a pick, places the foot of the prop in it, and pushes lid and prop as nearly into position as he can by hand, and finally drives it up with a hammer. The length of the prop should be such that considerable force is required to drive it up into position.

Chocks or Cogs.—When the pressure to be resisted is very great, chocks or cogs may be employed.

These are square pieces of timber built up two by two, placed crosswise. When used in the working faces they are built up on heaps of slack or dirt, so that they can be readily withdrawn by cutting away the support from underneath them. They yield at first to the pressure, but as the weight upon them increases, they become consolidated, and will then bear an enormous pressure.

Chocks or wood packs are also used in the gate-roads of longwall collieries, being built into the packs, more especially at the junction of one road with another.

Chocks vary greatly in size; the smaller are built up out of timber about 2 feet long, the larger may be constructed of old railway sleepers, or of broken props or bars.

The large chocks are often partly constructed of stone, which is packed up inside them.

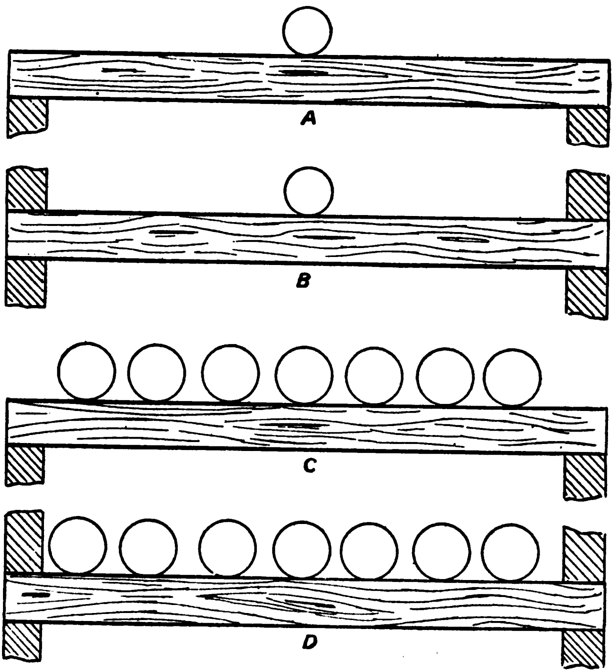
Bars.—A bar is exposed to a strain acting at right angles to its length; this is known as a transverse strain.

The strength of a bar may be calculated from the following formulæ:—

W = breaking load in cwts.
B = breadth of bar in inches.
D = depth ,, ,,
L = length ,, ,,
K = coefficient of rupture.

$W = \frac{4KBD^2}{L}$ when load is in centre and ends of bar are supported, but not fixed, as at A, Fig. 82.

$W = \frac{6KBD^2}{L}$ when load is in centre and ends are fixed, as at B, Fig. 83.



FIGS. 82, 83, 84, 85.

$W = \frac{8KBD^2}{L}$ when load is distributed and ends supported, but not fixed, as at C, Fig. 84.

$W = \frac{12KBD^2}{L}$ when load is distributed and ends fixed, as at D, Fig. 85.

The value of K differs slightly according to various authorities. Molesworth gives it as 12 for Memel and fir, 10 for yellow pine, and 15 for English oak.

The formulæ given above show that the strength of a bar varies—

(*a*) Directly as its breadth. Thus, if the breadth of a bar is doubled, the other dimensions remaining the same, the weight it will carry is also doubled.

(*b*) As the square of its depth. Hence, if the depth is doubled, the strength is increased fourfold, or in the proportion of 1^2 to 2^2.

(*c*) Inversely with the length. Therefore, if the length between the supports of a beam is halved, the weight it will carry is doubled.

(*d*) According to the manner in which the load is applied. A beam upon which the load is evenly distributed will carry twice as much weight as a similar beam which has the whole of the load applied at the centre.

(*e*) With the method in which the ends are secured. If the ends are fixed as at *a*, Fig. 86, the bar must break in three places before it fails; whereas if the ends are only supported as at *b*, the bar fails by breaking in one place.

These rules show that in fixing bars the following points should be kept in view: (1) The span should be no longer than necessary; (2) the deep side of the bar should be set vertically; (3) the bar ends should be well wedged to the roof; (4) the concentration of the weight on one part of the bar, as, for example, by wedging it tightly to the roof from the centre, should be avoided. All these points are frequently neglected in practice. The following examples show the loss in strength that may result from setting timber improperly:—

A pine bar is 10 inches by 14 inches in section: find its breaking strain—

(*a*) When set on its broad side, with a span of 8 feet, and having ends supported and weight concentrated in the centre.

(*b*) When set on edge, with a span of $6\frac{1}{2}$ feet and having ends fixed, and weight spread along its entire length.

$$(a)\ W = \frac{4KBD^2}{L} = \frac{4 \times 10 \times 14 \times 10 \times 10}{96} = 583{\cdot}3 \text{ cwts.}$$

In this case the formula for beams having ends supported and weight in centre must be used, and as the beam is laid on its broad side, its depth is 10 inches, which has to be squared.

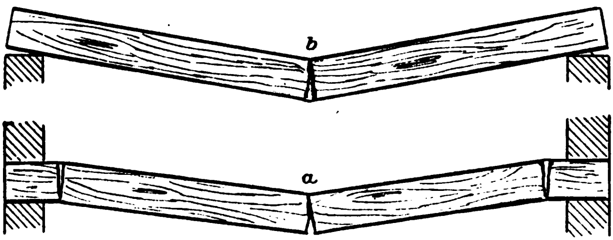


FIG. 86.—Fixed and supported beams.

$$(b)\ W = \frac{12KBD^2}{L} = \frac{12 \times 10 \times 10 \times 14 \times 14}{78} = 3015{\cdot}4 \text{ cwts.}$$

In this case the formula for beams with fixed ends and load distributed applies, and as the beam is set on edge, its depth is 14 inches, which has to be squared.

These examples show that by reducing the span and fixing the beam to the best advantage, the strength is increased from about 29 to about 150 tons.

Three methods of fixing bars are shown at A, B, and C, Fig 87. At A the bar is supported by the sides, which are notched to receive it. Bars of this description are known as "stretchers," and can only be employed when the sides are strong. With loose sides, bars supported by props, as shown at B, are set; if the roof is very loose, it is supported between the bars by "covering wood," which consists of timber slabs

placed longitudinally from bar to bar. "Herringbone" strutting is shown at C, Fig. 87. This form of timbering is largely employed in the main roads of some of the Midland collieries, when the sides of the roads are sufficiently strong to afford the necessary support to the struts. The balk *a* is long enough to enable several pairs of struts to abut against it, thus binding the whole together. The strength of timber set in this manner is due to the fact that the struts take the greater part of their load by compression and not transversely,

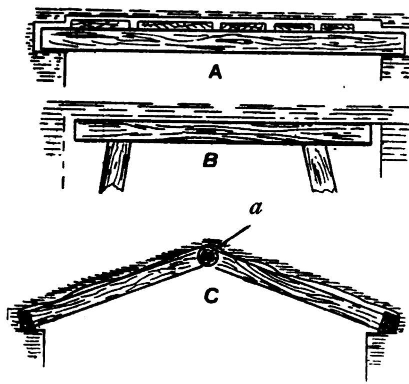


FIG. 87.—A, stretcher; B, bar; C, herringbone strutting.

and so act more like props than bars. Herringbone strutting looks well and is strong, but it is rather costly to set.

Sprags and Cockermegs.—During the process of holing the coal has to be supported by "sprags," their distance apart being fixed by the Coal Mines Regulation Act at not more than 6 feet.

In seams lying at a high inclination, and in some thick seams, sprags do not afford sufficient protection to the men engaged in holing, as the coal may burst off from the face in slabs. To prevent this, "cockermegs" are set. Fig 88 shows a sprag and "cocker" set in position. When the floor is hard, a hole should be stamped in it to receive the end of the sprag, which should be wedged tightly against the coal.

The cockermeg consists of a piece of timber 2 or 3 feet long (*a*, Fig. 88) set horizontally and wedged to the floor and roof by the sprags *b* and *c*.

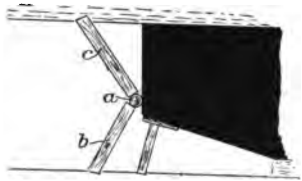


FIG. 88.—Sprag and cockermeg.

Timbering Longwall Faces.—The object of timbering longwall faces is to steady the roof, and keep a temporary road open along the faces.

The timber should be set at regular intervals, and systematically withdrawn when no longer required. The roof at the coal face is partly supported by the timber, and partly by the coal itself, and it is of the greatest importance to arrange the timber in such a manner that the coal has just sufficient weight upon it to cause it to work easily.

The packs only carry the gates, they do not assist in the support of the face, because they resist no great pressure until crushed and consolidated by the weight, and this does not happen until they are some distance back from the coal. The roof gradually sinks in the goaves as the face advances, the rate of subsidence amounting in some cases to about 1 inch per foot, so that 10 feet back from the face the roof has sunk about 10 inches. It is impossible to resist the enormous pressure which produces this subsidence, so that if props are left in the goaves, either they or the roof around them are broken, and extra weight is thrown upon the face and timber supporting it. For these reasons all props should be drawn from goaves, packs, and, wherever possible, the gates.

When working by longwall, two rows of props should be kept at the face, and the back timber regularly drawn, and the roof allowed to break behind the second row. When the roof is very bad, bankbars must be set in addition to the props, to prevent the roof from breaking down between the props.

The packs should be built entirely of hard lumps, as large as can be obtained; and special attention should be given to the gate-end packs, which are built from the material got from the ripping.

Tapered Props.—The great breakage of props which occurs at a longwall face, especially when the floor is hard, is due to the subsidence of the roof, which inevitably takes place as the face advances, the pressure which causes this subsidence being so great that no timber can withstand it. To reduce this breakage, Mr. Hepplewhite has recently introduced the employment of tapered props.

These are ordinary props, having one end tapered down to about half the original diameter, the tapered portion being about 12 inches long. As the weight settles on to a prop of this description, the tapered end burrs under the pressure as the roof subsides, thus saving the prop from being broken. A prop 6 inches in diameter begins to "burr" when the pressure upon it is about 16 tons, and breaks under the same pressure as an ordinary prop of the same diameter, that is, about 40 tons. After a prop has been set, and the end become burred, it is sent out of the pit to be re-tapered; when it can be used again, either for a thinner seam, or it may be used for the same seam, being set with an additional lid.

Steel Girders.—Steel girders of H section are now frequently used instead of timber bars, for supporting the roof in main roads; they are more costly in the first instance, but last much longer, and carry a greater weight with less loss of head room. They are also used in a few places for props, but they have not hitherto been adopted very widely for use at the coal-face, as they are apt to get lost, and buried by falls of roof. The ends of steel girders used as props should be flattened; this is usually done by cutting a piece out of the web at each end, and bending the flanges over till they meet and form a flat surface. Steel props should always be set with soft wood lids on roof and floor. Cast-iron props have been tried at several places, but have not met with success, as sudden strains cause them to break without warning.

A steel prop of H section 5 inches by 4 inches, weighing 50 lbs. per foot, and 5 feet in length, has a breaking strain of nearly 100 tons; and a similar prop, 4 inches by $3\frac{1}{2}$ inches, weighing 38 lbs. per foot and 4 feet long, breaks with a pressure of about 70 tons.

The ordinary sections of steel girders, when used for bars, are as follows:—

5 in. deep,	4 in. flanges,	$\frac{3}{8}$ in. thick,	weight	50 lbs.	per	yard.
5 ,,	4 ,,	$\frac{1}{2}$,,	,,	65	,,	,,
6 ,,	$4\frac{1}{2}$,,	$\frac{1}{2}$,,	,,	72	,,	,,

Steel girders are usually set in a similar manner to timber bars, the strongest position being, of course, with the web vertical. They may be let into the sides when these are sufficiently strong to carry them, or they may be supported by props, or by side walls of masonry.

When set on props, wrought-iron lugs may be shrunk on to the girders a few inches from either end, to prevent them from slipping.

They may be prevented from canting by having boards driven in between the flanges from girder to girder.

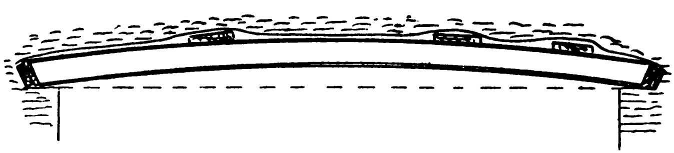


FIG. 89.—Steel girder, set with camber.

Steel girders will bend considerably before breaking, and when bent can be readily straightened and used again. If the deflection is not serious, the bent girder may be reset without being straightened, but should be turned the other way up, so that the result of the weight is to straighten it out. Girders set as bars are subjected to a transverse strain, but their carrying capacity can be very greatly increased by setting them with a certain amount of camber, as shown in Fig. 89; the usual amount of camber is about $\frac{1}{20}$ of the span. The ends of girders set in this manner must be firmly wedged up with hard wood, and the sides of the roadway in which they are set must be very strong.

Steel girders are often set "herringbone" fashion, the

longitudinal centre-piece against which they abut being of timber.

Masonry.—Pit-bottoms and important main roads are frequently lined with brickwork ; brick arches, though costly, making the best permanent lining that can be obtained, but of recent years girders placed upon brick side walls have been often used as a substitute.

If girders become bent or broken, they can be replaced without much difficulty, but if an arch becomes crushed, it has to be taken out and rebuilt, which is rather a serious matter. The simplest form of arch consists of two vertical side walls carrying a semi-circular, or elliptic arch ; but this construction is weak, as the side walls can be pushed in with a moderate pressure. All arches should be constructed so as to be subjected to a compressive strain ; a perfect circle would be the strongest form of arched road, but could not be employed owing to the large area of waste space. Unless the floor is very strong, an inverted arch should be turned at the bottom of the road, which, together with the main arch, forms a complete lining of brickwork. The brickwork should on no account be set to touch the strata, but should be separated from it by a space of a few inches, which should be tightly packed with sand.

Fig. 90 shows an arch suitable for a double road. It will be noticed that it has no straight walls, and that the result of weight upon it would be to compress the bricks, which must be crushed before the arch can give way. The crushing strain of good hard red bricks is about 3000 lbs. per square inch. Courses of wooden blocks, similar in size and shape to ordinary bricks, are sometimes built into arches ; being softer than bricks, they are compressed before sufficient pressure is applied to the arch to break the bricks.

To put in a length of arching similar to the one shown in Fig. 90. The ground is first got out for a short length, the main timbers used to temporarily support the excavation being arranged longitudinally instead of across the road. The floor is then dressed with picks to the shape of the invert, and a

layer of sand a few inches thick is laid upon it. The brickwork is then laid upon the sand, being kept in the correct shape by means of a wooden template.

The brickwork is carried up in independent rings; thus an arch 14 inches in thickness would be constructed of three separate rings of bricks with no bond between them. After the invert is laid, the main arch is carried up, the sides being built by the aid of a template, and the crown upon centres. The centres may be constructed either of timber or iron; in the figure, one of the iron centres, *a*, is shown in position. The laggings *b*, *b* are carried by the centres, and the brickwork is built up on them. All the rings of brickwork must be

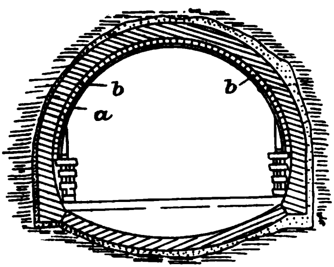


FIG. 90.—Arch for double road.

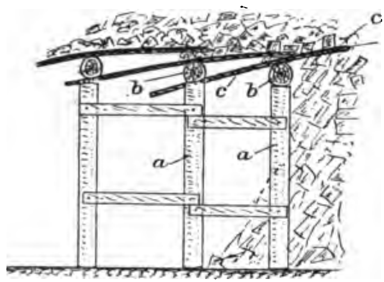


FIG. 91.—Spiling through loose ground.

carried up from both sides simultaneously to equalize the weight upon the centres, the inner rings being kept a course or two in advance. As the top of the arch is reached, short laggings, known as blocking lags, are fixed at right angles to the main lags, and the few courses at the crown of the arch are built up on them. The whole of the timber used to support the excavation must be drawn out as the arch is built, and the space between the walls and strata should be filled in with sand. Arches should not be put in unless the road is perfectly settled, otherwise they may be broken and have to be taken out and rebuilt.

Timbering through Loose Ground.—Special means are required to drive roads through very loose ground. Fig. 91

shows the method of spiling or poling which is usually adopted for passing through very loose or wet ground, such as is sometimes met with when scouring through faulty ground, goaves, or very heavy falls. *a*, *a* are props carrying the bars *b*; over the bars, the boards *c* are driven in advance, and are 5 or 6 feet long and 1 inch thick. These boards are driven into the loose material, as shown, a portion of it is excavated, and another prop and bar set.

The Courrières System of Timbering.—This method of timbering is in use at the large Courrières collieries, which are situated near Lens, in France.

The death-rate from falls of ground at Courrières during the ten years prior to 1899 was 0·15 per thousand persons employed underground, whereas in Great Britain the death-rate was 0·79 per thousand during the same period. The attention of the British coal-owners was called to this, in 1900, by a circular letter from the Home Secretary, and the question of adopting a similar method in England was discussed. The chief features of the method are: (1) The whole of the working places are timbered systematically; (2) each workman is supplied with three iron bars, each about $1\frac{3}{8}$ inch square and 4 feet 3 inches long, which he drives over the last bar to form a temporary shield in advance of the timber.

The main bars are set in rows parallel to the face, the rows being 3 feet 9 inches apart. Each of these main bars is about 8 feet long, and is supported by three props. Smaller bars rest on these main bars, being fixed at right angles to them, and spanning the space between the rows. These smaller bars are placed about 12 inches apart. When the workman has cut away about 12 inches of coal in front of him, he drives the iron bars over the main bars towards the face, to form a temporary support.

The iron bars are held in position by wedges from the back, and are driven forward as the face advances, being never more than 8 inches behind the coal. By this method every square foot of the roof is always supported.

The cost of timber is extremely high, as it is seldom drawn from the goaves.

The difficulties of introducing this system into England—apart from its prohibitive cost—would be enormous, as it would necessitate the altering of the whole method of work. It would also render holing by machines, or even extensively by hand, impossible.

There are many English collieries, timbered in the usual manner, which show quite as favourable a death-rate as the Continental collieries timbered in this way.

CHAPTER XVI.

COAL-CUTTING BY MACHINERY.

THE employment of coal-cutting machines has increased very greatly during the last few years; this has been brought about by the scarcity of labour and the approaching exhaustion of the best seams at many collieries. The following statement shows the number and type of the machines now in use in Great Britain (1903):—

Tons of coal got by machines (1903)	4,161,202
Total number of machines in use	483
Machines driven by compressed air	334
Machines driven by electricity	149
Disc machines	395
Pick machines	29
Revolving bar machines	21
Rotary heading	25
Chain machines	10
Class not stated	3

Over one-half of the total output of machine-cut coal was obtained from Yorkshire and the Midlands.

At many collieries coal-cutting machines have been introduced and abandoned after a short trial, whilst at other places they are largely used, and are an unqualified success. Their success or failure depends upon the suitability of the seam for machine work, and possibly even more upon the skill and resolution of the management. Like all innovations, much trouble is usually experienced in the early stages, and it sometimes happens that they are given up before these initial difficulties have been overcome.

Coal-cutting machines may be divided into three classes—

1. Machines designed to hole the coal in longwall faces.
2. Heading machines.
3. Machines that can be used either for headings, for longwall faces, or for short banks, as in pillar and stall.

Machines for holing Longwall Faces.—These machines have the greatest chance of success when working in thin seams which are expensive to get by hand. If the getting price of hand-cut coal is low, little saving in cost can be effected by machines, though they may be of advantage by increasing the proportion of round coal. It is desirable, though not absolutely essential, that the roof should be good and the inclination not excessive. The roof is often found to improve when the coal is worked by machines, owing to the quicker and more regular advance of the faces. The seam should be fairly free from faults, as a considerable length of clean face is necessary for efficient working.

Advantages of Machine Holing.—The advantages attending the employment of holing machines, under suitable conditions, are—

1. More round coal is made. Under average conditions about 10 per cent. more round coal is made. This may be a matter of the greatest importance, but, if the slack is of value for coking and other purposes, the advantage may not amount to much. The increase in round coal amounts to much more than 10 per cent. in some cases, as, for example, when a seam is thin and hard, and holing has to be done in the coal by hand, whilst by machines it can be done in the under-clay, or in a dirt band.

2. The cost of getting the coal is somewhat reduced. The reduction in cost, under favourable conditions, averages about 6*d.* per ton, or probably less when consumption of steam, interest on capital, and depreciation of plant are carefully accounted for. The cost depends upon the nature of the seam, and upon the bargain which can be made with the men.

3. The output per man employed is increased. In many

cases the output per man is about doubled, the proportionate increase being greater in thin seams.

4. The length of face required for a given output is lessened. This reduction in pit room leads to a reduction in the cost, as the workings can be concentrated and the length of roads reduced.

Description of Machines.—The machines cut a groove in or under the coal, and at the same time propel themselves along the face. The cut is made by means of chisels fixed either in a disc, bar, or chain.

Disc Machines.—Machines of this class consist of a frame

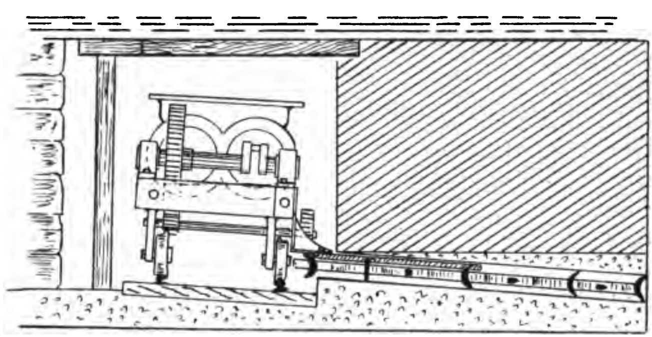


FIG. 92.—End view of disc machine.

mounted on small wheels carrying a pair of engines or an electric motor. A bracket projects from the machine, and carries the cutter wheel or disc, which has chisels set in its rim. Bevel teeth are cast in the wheel, and engage with a small bevel pinion driven by the engine through spur gearing. Fig. 92 shows an end view of a disc machine as it appears when making a cut. Whilst the disc revolves, and the machine cuts, it pulls itself along the face by means of a light steel rope. This rope passes from a drum on the machine round a pulley attached to a prop set in advance, and back to the machine, so that as the rope is wound on the drum the machine is pulled along. Arrangements are always made for varying the

speed of the machine's advance to suit the material it has to cut. Fig. 93 shows one form of propelling gear. *a* is a spur wheel, driven from another spur wheel, *b*. *a* also serves as a crank with adjustable throw. The crank pin is carried by a slot, along which it can be moved. In the figure the crank pin is at the end of the slot, and has its maximum throw. To reduce the length of the crank, the pin is moved along the slot nearer to the centre of the wheel. The connecting rod *e* drives the ratchet lever *c*, and, as the ratchet lever is much longer than the crank, the lever vibrates to and fro, but does not rotate. *d* is a ratchet pawl, which drives the ratchet wheel *k*. As *c* vibrates it catches the teeth in the ratchet wheel and pulls it round, whilst the check pawl *f* prevents the ratchet

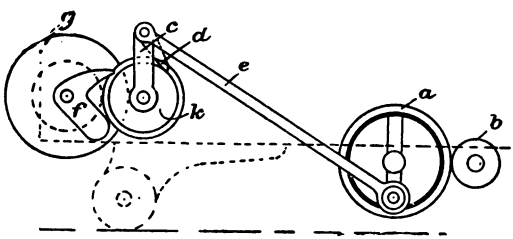


FIG. 93.—Propelling gear for coal-cutter.

wheel from running back. If the throw of the crank is at its maximum, as shown in figure, the pawl catches several teeth and the drum revolves at top speed; but, if the throw is reduced, the length of stroke of the ratchet is also reduced, so that it does not catch as many teeth in the wheel, and therefore pulls the drum round more slowly. The drum *g* is driven by spur gearing on the ratchet wheel.

Depth of Cut.—The depth of cut varies from about 2 feet 9 inches up to 7 feet. Deep cuts are of advantage in some seams, as fewer cuts are required for a given output, and in some cases the number of shots is reduced. It is also claimed that, by having deep cuts, the roof at the face is not broken, as the distance between the breaks is equal to the depth of the

cut. In some seams, if deep cuts are taken, the coal sinks and the floor lifts until the cut is practically closed; when this is so, the labour involved in breaking the coal up may be almost equal to the labour of getting the coal by hand and the number of shots required to break up the coal increased. The usual depth of the cut is from 4 to 5 feet.

Width of Cut.—The width of the cut varies from $3\frac{1}{2}$ inches up to 6 or 8 inches. In coal the width should be small, otherwise a considerable percentage of the seam is ground up into slack. Sometimes the seam has a band of dirt in which the machine can be arranged to hole. When this is so, the width of the cut is modified to suit the width of the band. When there is no such band, and the seam is thin, the holing should, if possible, be done in the underclay. Some machines will hole on the floor level, in which case the cut is made horizontal, but, if the machines are not adapted for holing on the floor level, the machine should be tilted as shown in Fig. 92, otherwise the "bottoms" will have to be got up by hand all along the face.

The following are brief descriptions of some of the principal disc machines:—

The Diamond Machine.—Fig. 94 shows this machine as arranged to be driven by electric motors. Two motors are employed, one being fixed at either end of the machine, and the disc carried by a bracket in the centre. About ten bosses are cast into the periphery of the disc, the cutters fit into boxes, in groups of three, and the boxes are secured to the bosses by a pin. This cutter-box arrangement is a speciality of the diamond machine, and has greatly reduced the time required to change the cutters. The weight of a diamond machine is from 25 to 45 cwts., according to the depth of the cut it is capable of making. These machines are arranged to work in either direction, that is, along a face and back again. They are made for very deep cuts, and are very strong and well suited for holing in hard ground. A fair shift's work for a machine, when holing to a depth of 4 feet 6 inches in hard ground, is from 50 to 60 yards. Three

men are employed to work the machine. When driven by electricity, two motors are employed, one being fixed at either end. The disc revolves at about fifteen or twenty revolutions per minute.

Rigg-and-Meiklejohn.—This machine is more suitable for holing in coal than in underclay.

The disc revolves very rapidly (from sixty to eighty revolution per minute), and is provided with bosses to carry twelve cutters, but in some cases only six are employed. When so few cutters are used, and the speed is so rapid, the effect of

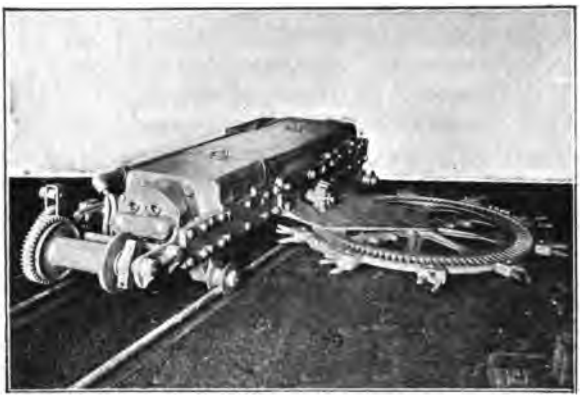


FIG. 94.—Diamond coal-cutting machine.

the cutters is percussive, that is, they strike a series of blows, whereas in most machines the action is more like that of a saw.

Machines of this class are driven by compressed air. Each machine has a pair of cylinders about 8 inches in diameter. They are placed side by side at one end of the frame, and the disc is carried by a bracket at the other end. The propelling gear consists of a ribbed drum driven by gearing from the engine. A $\frac{3}{8}$-inch chain is secured to a prop in advance of the machine, and makes three or four turns round the drum.

The man who holds the chain in his hand can regulate the speed of advance whilst the speed of the drum remains constant, by slackening the chain and allowing it to slip on the drum.

The bearings in which the wheels run are carried by screws, which can be raised or lowered at will, to enable the disc to adapt itself to any inequalities in the floor. The cut is usually from 2 feet 9 inches to 3 feet 6 inches deep, and the advance is extremely rapid, over 200 yards per shift being holed at some collieries. It is a very simple and strong machine, can run both ways, and for holing in the coal is equal to any machine on the market.

The Gillot-and-Copley Machine.—This machine is arranged to work by compressed air. It has two cylinders fixed side by side at one end of the frame, and the disc is carried by a bracket at the other end; the depth of cut is usually from 3 to 4 feet. The propelling drum is driven by a ratchet-wheel; the rope from the drum is taken round a pulley fixed to a prop set in advance, and back to a bridle fixed to the front of the machine. This bridle carries two pulleys, which run on each side of the outer rail and resist the thrust of the machine. The disc is fitted with about 25 teeth, and makes six or seven revolutions per minute.

These machines weigh about 20 cwts.; they make the cut above the level of the rail.

The Jeffrey Disc Machine.—This machine is made to be driven either by electricity or compressed air. The engines or motors are at one end of the frame, and the disc at the other. The special features are—

1. It runs on one rail only, the thrust being taken by spragging the rail to the roof with a light screwjack. By having only one rail, more room is available for shovelling away the *débris*.

2. It cuts on the floor level. (Several other machines are arranged to do this.)

3. The disc may be adjusted by a hand-wheel.

4. The advance motion can be stopped or started without

stopping the disc. This is a great advantage, as, when the disc jams, the feed can be stopped, thus giving the disc a much better chance of clearing itself.

5. Three rates of cutting are provided, 8, 16, and 25 inches per minute.

6. The machine is driven from the front, and will run either way.

Clarke-and-Steavenson's Machine.—This is an electrically driven machine, and is largely used in Yorkshire and elsewhere. A steel frame carries a motor at one end and the disc at the other. The disc is driven through spur and bevel gearing; the depth of cut is from about 3 feet to about 5 feet. The motor is of the enclosed type, and of 25 to 30 horse-power, and the weight of the whole machine is about 40 cwts.

Comparative Advantages of Disc Machines.—At the present time disc machines are by far the most popular. The table on page 190 shows that more than 80 per cent. of the machines of all classes working in Great Britain in 1903 belonged to this type. Notwithstanding the fact that they are used to such an extent, disc machines have considerable disadvantages. The discs take a large amount of driving power, the stuff made by the chisels is dragged round with the disc, greatly increasing the friction; this is especially the case when the machine is holing in a band of coal or dirt which is rather wider than the cut. For example, if the cut is 6 inches wide, and the stratum in which the cut is made is 10 inches thick, it is probable that the extra 4 inches will be dislodged, and the additional material greatly impede the rotation of the disc. When the coal is tender, large masses may be shaken by the vibration and fall or "sit" on the disc, causing frequent delays, and even rendering machine cutting impossible. Discs, too, are very heavy and bulky, so that, when the machines have to be "flitted" from one end of the face to the other, the cost and inconvenience is considerable.

Large discs are always made in halves bolted together, so that they can be removed in two pieces. In some types of

disc machines the wear of the pinions and other moving parts is very heavy, and may add considerably to the cost of holing. The rails upon which the machines travel along the face have to be very carefully set and spragged, as there is considerable thrust upon them. Disc machines, as a rule, do not cut their way into the coal, but require "wheel-holes" to be made at either end of the face to admit the disc at the commencement. These "wheel-holes" usually take the form of headings kept a yard or two in advance of the face.

With the diamond machine there is a special arrangement by which it can cut its way into the coal. The machine is placed on cross rails and set at an angle to the face, the disc is revolved, and the machine gradually pushed up to the face by screwjacks placed under the rails. The last yard or two at each end of the face has to be holed by hand, as the frame projects beyond the disc at either end.

Longwall Bar Machines.—These are similar in general design to the disc machines, except that the disc is replaced by a bar. This bar is studded with small chisels, and revolves at a high speed. As the machine pulls itself along the face the chisels revolve and cut a groove in the coal or under-clay.

The Hurd Machine.—This machine consists of an electric motor mounted on a frame and wheels, driving a bar, together with the usual propelling gear. The bar is so arranged that it can be drawn out of the cut for examination and to change the chisels; it can also be turned right over to enable the machine to make its cut higher up in the coal. The bar makes between four and five hundred revolutions per minute; it has a spiral groove turned along it to act as a conveyor for bringing the *débris* out of the cut. Between thirty and forty chisels are fitted into the bar, which, in addition to the rotary motion, has a reciprocating movement of 2 inches in and out of the cut. This prevents the bar from clogging, and helps to clear the cut. The cut is taper, and not parallel, as is the case with disc machines; it is usually 6 or 7 inches wide at the front, and 3 or 4 inches at the back. The chisels can be either set in the

groove or on the spiral, so that the width of the cut can be slightly varied. The speed of the drum can be varied by a cam acting on the rachet pawls.

The bar can be swung through a horizontal angle of 180 degrees without throwing it out of gear. Owing to this arrangement the bar can be made to cut its way into the coal at the commencement of the holing.

These machines are made in the following sizes:—

1. To cut up to 3½ feet in depth, weight about 20 cwts.; H.P. of motor, 12.
2. To cut up to 4½ feet in depth, weight about 30 cwts.; H.P. of motor, 18.
3. To cut up to 6 feet in depth, weight about 45 cwts.; H.P. of motor, 26.

The Lee Machine.—In this machine the cutting tool consists of a steel band wound spirally around a taper bar. The band has forty or fifty teeth cut in it, the advantages claimed being that the band is much more quickly changed than a set of chisels, and forms an efficient conveyor to clear the *débris* from the cut. Another speciality is the manner in which the machine propels itself along the face, neither drum nor rope being employed. It runs on two rails, one of which is in the form of a rack; a toothed wheel on the machine engages with this rack and drags the machine along. The rails are kept in position by being spragged from the roof. This machine can cut its way into the coal, and the bar can be raised or lowered within limits, to make the cut in the position best adapted for the seam in which it has to cut.

Comparative Advantages of Bar Machines.—Although bar machines are not very greatly used at the present time, they possess several advantages over machines of the disc type, and their employment will probably become more general. The bar is light and handy, and no great area is exposed to friction; hence bar machines require less power to drive them, and are not subjected to the great variations in load which are found in disc machines. Sprags can be set close up to the edge of the cut, and the machines cut their own way into the coal, so

that no heading is required to start them. Less gearing is required, as the bars revolve very rapidly.

The bearings of the bar are at the machine, and can be very efficiently lubricated; whereas the bearings of discs are actually in the cut, and so cannot be kept properly lubricated, and are exposed to great strain when the coal falls on the disc.

Bars are apt to cut unevenly. If the strata below is hard, they sometimes "climb" into the softer material, out of the proper level. They do not always bring the *débris* out of the cut very well. The reciprocating motion in the Hurd machine was partly designed to overcome this difficulty.

Longwall Chain Machines.—In these machines the holing

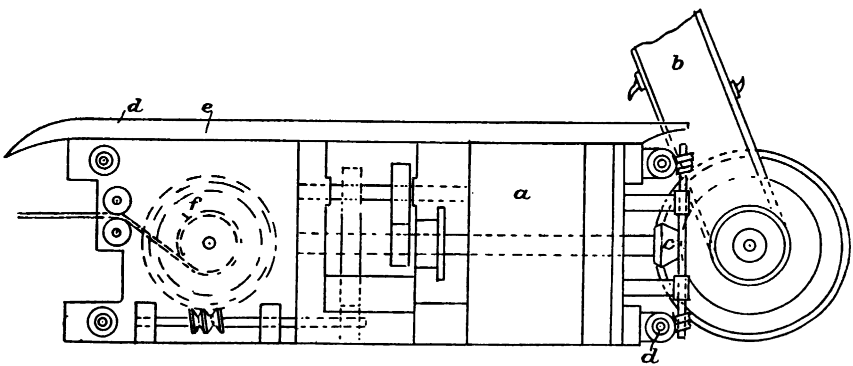


FIG. 95.—Morgan-Gardner coal-cutting machine.

is performed by cutters attached to the links of a chain. The oldest form is Baird's, and though this is now quite obsolete, other machines of a somewhat similar type—though differing very greatly in detail—are now being introduced.

The Morgan-Gardner Machine.—This is a new machine, which cuts the coal by means of chisels attached to a chain. Fig. 95 shows the machine in plan: *a* is an electric motor of about 12 horse-power; *b* is the chain jib which is attached to the back of the machine; it consists of two steel plates, kept apart by distance pieces; the endless chain runs between these

plates, and is driven by the sprocket wheel *c*, which is driven by the motor, and passes round a wheel at the end of the jib. The jib is free to swing right round the end of the machine, this movement being controlled by gearing (not shown in the figure) to enable the machine to cut its way into the coal. The machine does not travel along the face on wheels, but upon a pair of iron plates or sledges. It can be raised or lowered on these sledges by means of the four screwjacks, *d*, *d*, which are worked by worm-wheels driven by the motor. In order to keep the machine up to the face, the jib is set at an acute angle to the frame when making a cut, as shown in sketch. The iron fender *e* prevents the machine from being dragged into the coal. This arrangement does away with the trouble of laying rails along the face, and saves a couple of inches of height, but more power is required to pull the machine along. The newer machines are being made with a plough blade to take the outer thrust by cutting a groove in the floor. The propelling gear consists of the vertical drum, *f*, and guide wheels. The end of the rope is led round the fixed pulley and back to the jib, so that the cutters have a direct pull against the coal. The drum is driven through a friction clutch, and can be thrown out of gear without stopping the chain. The cutters are secured to the chain by set-screws, and are splayed out to make the cut sufficiently wide to admit the jib. The ordinary width of the cut is about $4\frac{1}{2}$ inches.

Working Coal by Machines.—There are two general principles upon which the working of coal-cutting machines can be arranged.

1st. By having a considerable length of face along which several machines work, cutting from one end to the other, and being "flitted" or moved back when they reach the end of the face.

2nd. By having a short length of face for each machine, along which it travels backwards and forwards, cutting both ways, the whole length of face being cut every night, and all the coal filled out during the day.

The first of these methods is the more generally adopted, the

workings being laid out as in ordinary longwall, except that the face must be absolutely straight, and a good road provided for "flitting" the machine from one end to the other. An ordinary cross-gate serves the purpose, but it should be laid out with a view to providing a short and easy journey for the machines. One advantage of this method is that the fillers can be kept a little in advance of the machines, so that there need be no delay on account of the coal not being got out of the way of the machine. Another advantage is that the cut can be made of any desired depth, and, as two or three shifts can be spent in getting out the coal, the gates can be set out a fair distance apart.

In the second, or short-face method, more coal is obtained from a given length of face, hence the workings can be concentrated and the faces advance very quickly; but the gates must be close together and the cut not very deep, otherwise it will be found impossible to get the whole of the coal out during the day shift. The cost of ripping the gates is very high by this method. In practice it is found that the gates must not be more than about 15 yards apart, and the holing not more than about 3 feet in depth in a seam 3 feet in thickness. If this system is to succeed, the discipline must be very good, and arrangements must be made for both fillers and machine men to stop at their work until all the coal is got out on the one shift and all the face cut on the other.

Timbering.—Machines require a width of about 4 feet from the face clear of props.

If the roof is good, props are set in rows, the first row being about 4 feet from the face. Each prop is provided with a long lid or plank to reduce the width of unsupported roof. If the roof requires the support of bars, one end of the bar rests on props, and the other is let into the coal, as shown in Fig. 92. When the coal is not sufficiently strong to carry one end of the bars, longitudinal bearing bars are set against the face to carry the ends of the short bars instead of the coal. The props which carry these bearing bars are removed immediately in front of the machine and reset behind it.

Cutting the Coal.—From two to four men are required to work a machine, three being the usual number.

The work of the machine men is to start and stop the machine when required, see that it is properly lubricated, and not subjected to undue strain, change the cutting tools when they become blunt, lay the rails and sprag them properly into position, set the timber at the face and clean out the cut, arrange the haulage rope, look after the hose-pipe or cable and make the connections when required, and attend to the many small details upon which the success of machine-cutting so much depends.

Three or four lengths of rails are required, the back length being taken up, handed over the machine, and relaid in front. As there is considerable outward thrust on disc machines, the rails should be of strong section, and securely stayed in position. The simplest form of rails are steel bars $1\frac{1}{4}$ to $1\frac{1}{2}$ inches square; holes are bored through each end, and are dropped on to pins fixed on the flat steel sleepers. Flat-bottomed rails weighing from 20 to 36 lbs. per yard are commonly employed; they are carried by sleepers made of dished steel fitted with projections to hold the bottom of the rails.

Motive Power.—Coal-cutting machines are driven either by electricity or by compressed air; at the present time compressed air is the more common motive power, but electricity is rapidly gaining ground. The advantages of electricity are that it is the more efficient, and therefore the more economical in steam; and that the cables along which it is conducted are more convenient than air-pipes. On the other hand, compressed air has advantages which go a long way towards making up for its lower efficiency, and more inconvenient conductors. It is absolutely safe in an explosive atmosphere—which electricity is not. The engine is cheaper, lighter, and simpler than the motor; moreover, it will stand rough usage better, and is more readily repaired by the men and with the appliances usually found at collieries.

Cost of Machine Holing.—The cost of the machines varies from about £250 to about £450 each, including accessories,

machines driven by compressed air being the cheapest. The cost of cutting is between 2½*d.* and 6*d.* per ton; in some places the cutting is let by contract, and at others the machine men are paid by the shift. The cost of steam, interest on capital, depreciation and repairs, amounts to about 6*d.* per ton on the coal cut.

Heading Machines.—The only machine now used exclusively for heading is the Stanley heading machine. It consists of a narrow frame carrying a pair of vertical engines which drive a central shaft through suitable spur gearing. The shaft carries a heavy casting upon which a pair of arms are bolted, cutters are attached to the arms, and, as the shaft is revolved by the engines, an annular space is cut around a central core, making a perfectly circular road. The core is wedged down and removed by hand in some of the machines, but in others it is cut up by the machine, and the pieces are elevated and deposited in the pit tubs by a conveyor. The central shaft, which carries the arms and cutters, has a thread cut upon it, so that it is slightly advanced at every revolution. The machine is held up to its work by screws which are forced into the roof, and, for ease in moving back, it is carried on wheels. In thin seams duplex machines may be employed, which cut out two circles side by side; or a single machine may cut out one circle, and another follow it up and cut out another circular road alongside the first, the triangular pieces next roof and floor being subsequently removed by hand.

These machines will drive a heading at the rate of about 1 yard per hour, but for regular work the rate of progress is about 4 to 6 yards per shift.

Machines for Use either for Headings or Banks.—Machines of this class are largely used in America, where the pillar-and-stall method of work is almost universal. They are also used to a limited extent in England for driving headings, and in a few places for getting the coal in pillar-and-stall workings, or in longwall faces.

The Jeffrey Chain Machine.—This is a breast machine, making its cut straight in front, and being moved along by hand after each cut. An outline of the machine is shown in Fig. 96. The bed frame *a* is stationary, and is held in position close up to the face by the screw *b*. The sliding chain cutter frame *c* works between the side girders of the bed frame, and is racked forward as the cut advances. *d* is the electric motor (air-engines may also be employed) which is attached to the sliding frame, and moves with it upon the bed-frame. The cut is made by chisels fixed to the endless chain. When the machine is making a cut, the sliding frame is gradually racked forward, and at the same time the endless chain which carries the chisels is driven round and round the

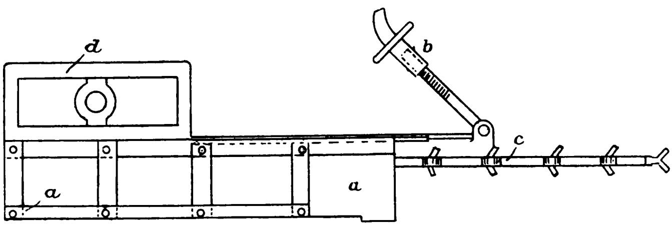


FIG. 96.—Jeffrey chain machine.

sliding frame, and so cuts its way into the coal. After the sliding frame has advanced its full length, it is run back by the engine, and the whole machine is moved along the face, and secured by screws ready for the next cut. The drawbacks to the employment of this machine for longwall work are its great length, making the timbering of the stalls very awkward; and the labour involved in sliding it along by hand. This latter is lessened by allowing the back of the machine to rest upon a rail or skid-board. For conveyance from one place to another, when used in headings or short banks, the machines may either be placed on trolleys, or mounted on wheels. Self-propelling trolleys are made, the wheels being actuated by the motor through chains. These machines are made to cut 5, 6, or 7 feet in depth, the width of each cut being

44 inches, and the height taken out by the cut about 4 inches.

Percussive Machines.—These machines are of American origin, the three types principally used being the Harrison, Ingersoll—Sergeant, and Yoch.[1] They have been tried at several collieries in Great Britain, but have not met with any great success. It has been found difficult to get men to work them; men new to the work suffer from the great vibration, but they are said to get over this with a little practice.

Fig. 97 shows a machine of this class as it appears when at work. It consists of an air-cylinder, *a*, from 4 to 6 inches in diameter, mounted on wheels, and provided with a pair of handles, *b*. The chisel *c* is attached to the piston-rod of the air-cylinder, and is driven rapidly backwards and forwards, striking a heavy blow forwards, and being forced lightly backwards against an air-cushion. The machines are really rock-drills mounted on wheels. When at work they are placed on sloping timber platforms (*d* in figure).

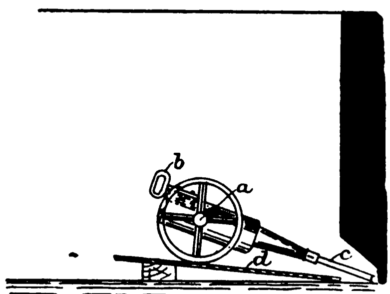


FIG. 97.—Ingersoll machine.

They are steered by the operator, who sits on the platform and directs the blows of the chisel with hands and feet. The recoil is taken by the slope of the platform, and, in some of the larger machines, by an automatic brake. The average amount undercut is about 50 square yards per shift. For making vertical cuttings a similar machine is employed, but it is mounted on larger wheels. The great advantage of these machines is in their portability, as they can very readily be moved from place to place. They weigh from 7 cwts. to 10 cwts., and cost about £100 each.

The Champion Coal-cutter.—This machine is very similar

to the percussive machines already described, except that it does not run on wheels, but is mounted upon a supporting column. It can be used either for shearing, holing, or drilling shot-holes, and, being light and portable, is suited for driving headings. Fig. 98 shows a plan of this machine as arranged for holing. *a* is the vertical supporting column, which is screwed tightly between roof and floor; it carries the machine by means of a sleeve, which can be fixed at any height upon the column. The toothed segment *b* is carried by a bearing on the sleeve, and is so arranged that it can be turned horizontally, as shown in Fig. 98, when holing is to be done,

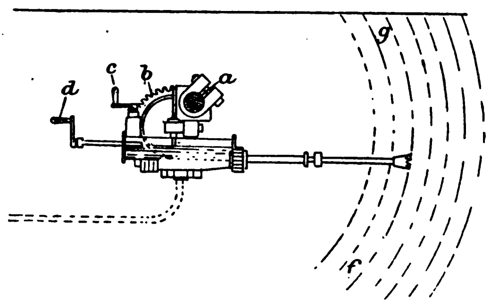


FIG. 98.—Champion coal-cutting machine.

or vertically when a shearing is to be made. The machine can be swung through the arc of a circle by turning the handle *c*, which acts on the segment *b*. The bit strikes about 350 blows per minute, and is advanced into the coal or stone by turning the handle *d*, which moves the whole machine forward on the column. After the machine is set up, compressed air is admitted into the cylinder, and the bit commences to strike the coal; as it does so, the handle *c* is slowly turned, causing the machine to gradually move round, so that instead of drilling a hole, it cuts a groove. After the machine has been swung through the width of the cut (*f* to *g* in figure), the feed-screw is advanced, and it is swung back again by reversing the handle *c*. Each cut is in the form of the segment of a

circle, as shown in figure. When the feed has reached its limit, the machine is run back, and an extension rod added between the piston rod and the bit. The width of the cut is about 3 or 4 inches, and the depth may be as much as 9 feet; machine and supporting column weigh about 5 or 6 cwts., and cost about £100.

CHAPTER XVII.

MECHANICS.

WORK is said to be done when a force acts through a space overcoming resistance.

The British unit of weight is the pound, and of length the foot, and the unit of work is the *foot-pound*, which is the amount of work required to raise a weight, or maintain a pressure of 1 lb., through a space of 1 foot, so that—

Foot-pounds = force exerted in pounds multiplied by space in feet through which the force is exerted.

Example.—How many foot-pounds of work are done in raising a ton of coal up a shaft 400 yards deep?

$$\text{Force} = \text{weight raised} = 2240 \text{ lbs.}$$
$$\text{and space through which it is raised} = 1200 \text{ feet}$$
$$\text{Then foot-pounds} = 2240 \times 1200 = 2{,}688{,}000$$

The power exerted in raising the above cannot be estimated unless the time occupied is known. Obviously, it would take a very much more powerful engine to raise the coal in 1 minute than in 10, hence *power* is the measure of the *rate at which work is done.* The measure of power is the number of foot-pounds per minute.

If the coal in the example given above were raised in $\frac{3}{4}$ of a minute, the power exerted in foot-pounds per minute would be $2{,}688{,}000 \times \frac{4}{3} = 3{,}584{,}000$.

In order to facilitate the expression of power without the use of such large figures as foot-pounds per minute, Watt

adopted a unit of power which he called the "horse-power," and this is now generally employed by engineers.

One *horse-power* is the power necessary to perform 33,000 foot-pounds of work in one minute, or—

$$\frac{\text{Foot-pounds per minute}}{33000} = \text{H.P.}$$

Hence the H.P. in the example given above is—

$$\frac{3584000}{33000} = 108{\cdot}6$$

The horse-power of a steam-engine is obtained by the following formula:—

$$\text{H.P.} = \frac{\text{P}.\text{L}.\text{A}.\text{N}}{33000}$$

where P = effective pressure in lbs. per square inch on piston.
L = length of stroke in feet.
A = area of piston in square inches.
N = number of strokes per minute.

The following example shows the manner in which this formula is arrived at:—

Find the horse-power exerted by an engine having a cylinder 18 inches in diameter by 3 feet stroke, when making 100 strokes per minute and having an average pressure on the piston of 45 lbs. per square inch.

The area of the piston is found by squaring the diameter and multiplying by 0·7854; thus the area is—

$$18 \times 18 \times 0{\cdot}7854 = 254{\cdot}47 \text{ sq. inches}$$

As the pressure per square inch is 45 lbs., the total pressure on piston is $254{\cdot}47 \times 45 = 11{,}451{\cdot}15$ lbs.

The length of stroke is 3 feet, and the number of strokes per minute is 100, so that the distance the piston moves per minute is $3 \times 100 = 300$ feet.

The *foot-pounds* exerted per minute are $11{,}451 \cdot 15 \times 300 = 3{,}435{,}345$, because the piston is subjected to a pressure of 11,451·15 lbs., and travels under that pressure through a space of 300 feet. As 33,000 foot-pounds per minute make 1 horse-power, the H.P. exerted is $\frac{3435345}{33000} = 104 \cdot 1$.

Condensing the above, we get—

$$\frac{45 \times 3 \times 254 \cdot 47 \times 100}{33000} = 104 \cdot 1$$

The energy transmitted by any machine is partly spent in doing useful work, and partly in overcoming frictional and other resistances.

The *modulus*, *useful effect*, or *efficiency* of a machine is the proportion of the energy expended which is converted into useful work and equals $\frac{\text{useful work done}}{\text{total work applied}}$. Efficiency may be expressed as a fraction, or, as is more usual, as a percentage. If, for example, an engine exerted 270 horse-power, and accomplished useful work to the extent of 180 horse-power, the efficiency would be $\frac{180}{270} = \frac{2}{3}$, or, expressed as a percentage, $\frac{180 \times 100}{270} = 66\frac{2}{3}$ per cent.

The horse-power developed in the cylinder of an engine is known as "indicated horse-power" (written I.H.P.), because the average steam pressure from which it is calculated is obtained by the aid of an indicator (see Chapter XVIII.). The actual horse-power available is known as the "brake horse-power" (written B.H.P.), and is arrived at by means of a dynamometer, or brake, arranged to absorb the power given off from a pulley.

The Mechanical Powers.—There are several mechanical appliances by which either force applied through a given distance is altered to a greater force moving through a correspondingly smaller distance, or they may operate in such a

manner as to alter the force transmitted into a smaller force acting through a proportionately greater distance.

In every case where W = force applied, D = distance through which the applied force moves, w = force transmitted, and d = distance through which the transmitted force moves—

$$W \times D = w \times d$$

In other words, neglecting frictional losses, the force applied, multiplied by the distance through which this force is exerted, equals the force transmitted multiplied by the space through which the transmitted force moves.

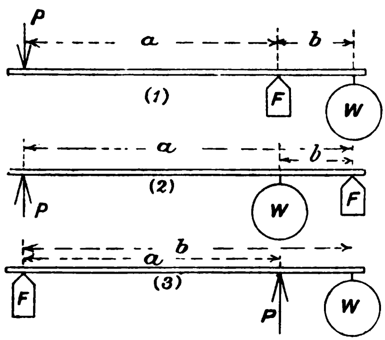


FIG. 99.—Levers.

The simple mechanical powers are: *The lever*, *the wheel and axle*, *pulleys*, *toothed wheels*, *etc.*, *the inclined plane*, *the screw*, *etc.*

Levers.—A lever is a rigid bar supported at one point called the *fulcrum.* The weight is placed at a certain distance from the fulcrum, and the power is applied at another point along the bar. Levers are of three kinds, and are shown at 1, 2, and 3, Fig. 99.

In the first kind, fulcrum F is between power P and weight W; in the second kind, weight is between fulcrum and power; in the third kind, power is between fulcrum and weight. In all three cases, where a = length of power arm, b = length of weight arm, P × a = W × b, because a is proportional to

the distance moved by P, and b is proportional to the distance moved by W.

The beam of an ordinary pumping engine is an example of a lever of the first kind. The power is applied by means of the pressure of steam on the piston, the axle of the beam is the fulcrum, and the weight is the weight of the pump-rods and water. The mechanical advantage gained by having one end of the beam longer than the other is found as follows: If the total pressure of steam on the piston is $8\frac{1}{4}$ tons, and the portion of the beam to which the piston is attached is 12 feet 9 inches long, and the other portion being 10 feet 6 inches long, what weight will the engine lift?

$$P \times a = W \times b$$
$$\therefore 8\tfrac{1}{4} \times 12\tfrac{3}{4} = \text{weight lifted} \times 10\tfrac{1}{2}$$
$$\therefore \frac{8\frac{1}{4} \times 12\frac{3}{4}}{10\frac{1}{2}} = \text{weight lifted} = 10\tfrac{1}{56} \text{ tons}$$

It will be noticed that the weight lifted bears the same proportion to the pressure applied as the length of the arm through which the pressure is applied bears to the length of the arm carrying the weight.

The most familiar example of a lever of the third class is found in the ordinary lever safety valve, in which the fulcrum is the fixed end, and the power is the pressure of the steam on the valve. The total pressure bearing down on the valve is due to the weight of the valve itself, and the weight of the lever and of the weight upon it. Fig. 100 shows an ordinary lever safety valve, all dimensions being given. The weight of the valve is 2 lbs., of the weight 60 lbs., and of the lever 10 lbs. Before calculating the effect on the valve of the weight of the lever, the centre of gravity of the lever must be determined. The centre of gravity of a body is a point within it upon which, if supported, the body will rest, or be balanced in any position; and the total weight of the body may be taken as acting at that point.

So that if the centre of gravity of the lever is 9 inches from the fulcrum, as shown in figure, the effect is the same as if its

whole weight were concentrated at that point. The pressure at which a boiler fitted with the safety valve shown in Fig. 100 would blow off steam can be calculated as follows:—

Pressure on valve due to the weight of the 60 lbs.	$= \dfrac{60 \times 24}{3} =$	480 lbs.
Add pressure on valve due to weight of lever	$= \dfrac{10 \times 9}{3} =$	30 „
Add weight of valve itself	=	2 „
Total pressure on valve ...		512 „

The diameter of the valve is $3\frac{1}{4}$ inches, so that its area is $3{\cdot}25^2 \times {\cdot}7854 = 8{\cdot}296$ sq. inches. Therefore, the pressure per square inch at which steam will blow off is $\dfrac{512}{8{\cdot}296} = 61{\cdot}7$ lbs.

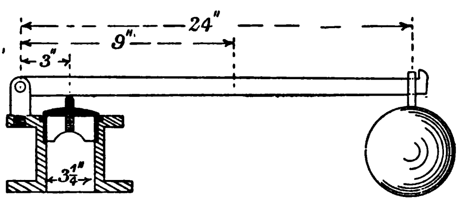


FIG. 100.—Lever safety valve.

Pulleys.—The wheel and axle and the pulley are merely forms of continuous levers, and the calculations relating to them are similar to the calculations relating to levers, the radii of the wheels representing the arms of the levers.

The most familiar example of the application of the wheel and axle is the ordinary windlass, in which the force is applied through a lever equal in length to the radius of the circle described by the handles, and the weight to be raised is hung from a lever equal in length to the radius of the barrel.

Example.—A windlass has a barrel 10 inches in diameter, and the circle described by the handles is 4 feet in diameter: what weight can be raised with three men at the handles, each exerting 26 lbs. pressure? In one revolution of the handles the

power travels through a distance equal to the circumference of a circle 4 feet in diameter, whilst the weight only moves through the circumference of a circle 10 inches in diameter; so that the weight raised

$$= \frac{\text{force applied} \times \text{distance}}{\text{distance moved by weight}} = \frac{3 \times 26 \times 48 \times 3{\cdot}1416}{10 \times 3{\cdot}1416} = 374{\cdot}4 \text{ lbs.}$$

By means of toothed gearing a greater difference in speed between the power and the weight can be obtained; thus a greater weight can be raised. Fig. 101 shows a single-purchase crab in which a small pinion on the handle shaft gears into a large wheel on the barrel shaft. In a double-purchase crab

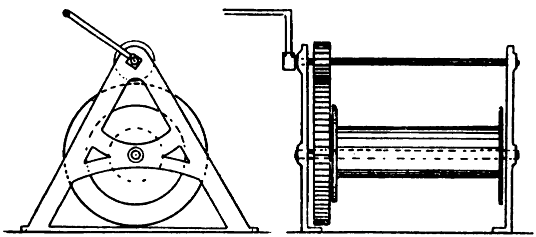


FIG. 101.—Single-purchase crab.

the speed is still further reduced by means of an intermediate shaft and gearing.

Belt pulleys are employed to drive machines, and by varying the size of the pulleys any desired speed can be obtained. Fig. 102 shows the manner in which an engine running at a comparatively low speed can be arranged to drive a high-speed machine, such as a dynamo or fan. The pulleys A and B are driving pulleys, whilst C and D are driven pulleys or followers.

To calculate the speed of the last wheel in a train.

When R = revolutions per minute of first driving wheel,
r = " " " last driven wheel,
D_1, D_2, etc. = diameters of driving wheels,
d_1, d_2, etc. = diameters of driven wheels,

$$\text{Then } R \times \frac{D_1 \times D_2}{d_1 \times d_2} = r$$

In the example given in Fig. 102, R = 80, D_1 = 10·5, D^2 = 8·75, d_1 = 3, and d_2 = 2,

$$\text{and } 80 \times \frac{10{\cdot}5 \times 8{\cdot}75}{3 \times 2} = 1225$$

so that the speed of the last pulley will be 1225 revolutions per minute.

When pulleys are connected by belts their speed varies with their relative diameters.

In the example given above, A is 10·5 feet in diameter and C is 3 feet in diameter, so that whilst A makes 1 revolution,

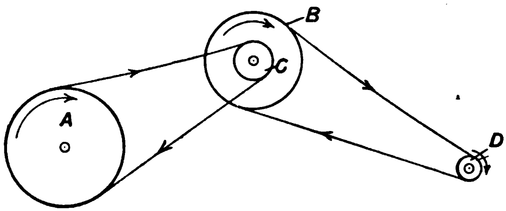


FIG. 102.—Belt pulleys.

C makes $\frac{10{\cdot}5}{3}$ = 3·5; hence C makes 80 × 3·5 = 280 revolutions per minute. B is keyed to the same shaft as C, so they both make the same number of revolutions; the diameter of B is 8·75 feet, and of D 2 feet, so that for every revolution B makes, D makes $\frac{8{\cdot}75}{2}$ = 4·375; hence, whilst B makes 280 revolutions per minute, D makes 280 × 4·375 = **1225.**

Toothed Wheels.—Belts or ropes and pulleys are employed when the speed of rotation is considerable; as, for example, in the driving of fans or dynamos. When a large amount of power has to be transmitted at a low speed, toothed wheels are superior; hence they are adopted for driving haulage gear, pumps, capstan drums, etc. The manner of calculating

the relative speeds of toothed wheels is similar to the manner described for determining the speeds of belt pulleys. The measurements must be taken along the *pitch circle*, which is the circle which passes through the points of contact of the teeth. The teeth are constructed in such a manner and of such a shape that they roll upon each other at the point of contact without any grinding action. For very exact calculations as to the relative speed of toothed wheels, it is better to take the number of teeth in each wheel rather than the diameters of the pitch circles. Then the revolutions of the two wheels are inversely proportional to their numbers of teeth.

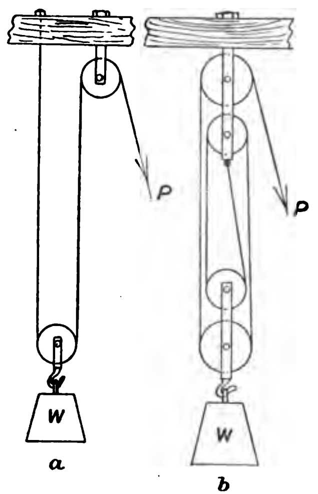


FIG. 103.—Block pulleys.

Block Pulleys.—Pulley blocks are appliances for lifting heavy weights, the mechanical advantage being obtained by passing a rope round a number of pulleys or sheaves. *a*, Fig. 103, shows a single sheave arrangement, by which the weight applied at P on the loose rope would (neglecting friction) balance double its weight when hung on the hook at W. The block is carried by two ropes, the pull on each of which is equal to that on P, so that the total weight balanced is double the weight on P. It will be noticed that whilst P moves 1 foot, W only moves 6 inches. The length of rope between the point of suspension and the fixed pulley is shortened by 12 inches, but the two ropes from which the weight is hung share this 12 inches between them. So that whilst the weight raised is double the pressure applied, the space moved through by the weight is only half the space moved through by the power. This is in accordance with the rule which applies to all the mechanical

powers:—That the force applied multiplied by its motion equals force transmitted multiplied by its motion.

By increasing the number of sheaves or pulleys in each set of blocks, the mechanical advantage can be multiplied to any extent.

In *b*, Fig. 103, two sheaves are shown in each block. The rope is fixed to the upper block, and taken round the pulleys as shown; in this case there are four ropes carrying the load, so that the power applied at the loose rope is multiplied by four, and the space moved through by P is divided by four. In practice the sheaves in each block are placed side by side on the same spindle to make the arrangement more compact.

The mechanical advantage of a set of pulleys equals the number of ropes leading from the bottom blocks. Taking the pull on rope at 150 lbs., in the arrangement shown at *a* (Fig. 103) the weight raised would be $150 \times 2 = 300$ lbs.; in *b*, $150 \times 4 = 600$ lbs.; in each case neglecting friction and the weight of ropes and bottom blocks.

The Inclined Plane.—The inclined plane is employed for raising bodies from one level to another. It may be inclined at any angle to the horizontal, and the inclination may be expressed either in degrees or by the relative proportions of height and base.

Thus, a plane is said to have a gradient of 1 in 10 when the rise or fall is 1 foot *vertical* for every 10 feet measured horizontally.

The power required to push or pull a body up an inclined plane, when the power is applied in a direction parallel to the plane, bears the same proportion to the weight as the height does to the length of the plane.

For example, find the strain on a hauling rope when pulling a load of 12 tons up an incline dipping 1 in 4. The strain on rope bears the same proportion to 12 tons as the length of the plane bears to the height, so that it will be $12 \times \frac{\text{height}}{\text{length}}$. The height and base are both given, but the length of the plane (hypotenuse) must be calculated. (For moderate inclinations

the length and base are so nearly the same that in practice the base is often taken instead of the length.)

The plane may be regarded as a right-angled triangle, the relative lengths of perpendicular and base being as 1 is to 4.

The relative length of the plane being the hypotenuse $= \sqrt{1^2 \times 4^2} = \sqrt{17} = 4{\cdot}123$, hence the strain on the rope is $12 \times \frac{1}{4{\cdot}123} = 2{\cdot}9$ tons.

A force of 2·9 tons is applied through a distance of 4·123, and results in a weight of 12 tons being raised through a space of 1, and $2{\cdot}9 \times 4{\cdot}123 = 12 \times 1$.

This calculation only shows the pressure required to overcome the gravity of the load; the friction must be worked out separately (see Chapter XXIV.).

Screws.—From a mechanical point of view, a screw is an inclined plane wrapped round a cylinder. Screws used in combination with levers may be used for lifting heavy weights, either in the form of the screwjack or in the form of lifting screws.

The pitch of a screw is the distance the nut travels along it at each revolution; thus, if a screw has a pitch of ½ inch, the nut upon it would advance ½ inch per revolution.

In the ordinary lifting screw, the weight is hung from the end of the screw, which passes through a nut carried by beams. The nut is revolved by means of a spanner or lever, which raises the screw and the weight it carries through the nut. The force exerted at the lever, multiplied by the circumference of the circle through which it acts, equals the weight raised multiplied by the pitch of the screw.

Example.—A force of 90 lbs. is exerted at the end of a spanner 3 feet 6 inches in length; the spanner acts on the nut of a screw having a ¾-inch pitch. What is the weight lifted by the screw?

The circumference of a circle 42 inches radius is $42 \times 2 \times 3{\cdot}1416 = 263{\cdot}8$ inches.

$$\therefore\ 90 \times 263{\cdot}8 = W \times \tfrac{3}{4}$$

$$\therefore\ W = \frac{90 \times 263{\cdot}8}{0{\cdot}75} = 31{,}656 \text{ lbs.}$$

In this, as with all the other mechanical powers, the gain in power is exactly proportional to the loss in motion. The friction of screws and nuts is very great, and is not taken into account in the calculation.

Hydraulics.—Water is a practically incompressible fluid, and transmits pressure equally in every direction. The *pressure per square inch* due to a column or head of water depends entirely upon the vertical head, and is independent of the shape or area of the column. Thus, if the water in a vessel is 5 feet deep, the pressure upon each square inch of the bottom is equal to the weight of a column of water 1 square inch in area and 5 feet long. This is not affected in any way by the shape of the vessel. It may be conical, round, or any other shape, but the pressure per square inch on the bottom is the same.

A cubic inch of water weighs 0·0362 lb., so the weight of a column of water 1 square inch in section and 1 foot long is $12 \times 0{\cdot}0362 = 0{\cdot}434$ lb. From this it follows that the pressure in pounds per square inch due to a head of water equals vertical head in feet $\times\ 0{\cdot}434$.

Example.—A shaft 1200 feet deep has a water-pipe the whole way down it. What is the pressure of water per square inch at the bottom?

$$1200 \times 0{\cdot}434 = 520{\cdot}8 \text{ lbs.}$$

If this water-pipe were connected to a boiler in the pit, and the boiler allowed to fill, and all outlets closed, the pressure per square inch on the whole surface of the boiler would be 520·8 lbs. If the column of pipes is not vertical, but is inclined, the pressure given rise to is that due to the difference of level between the two ends of the pipe; or, in other words, the vertical distance between the two ends.

Hydraulic Machinery.—The fact that fluids transmit

pressure equally in every direction is taken advantage of in the design of hydraulic rams and jacks. The pressure driving hydraulic rams may be either natural or artificial. In the former case, the pressure is obtained from a natural head; and in the latter case, from rams driven by steam or otherwise. Natural heads are sometimes employed in working hydraulic lifts in pit bottoms. A pipe is taken to a supply of water some distance up the shaft, and its bottom end is connected to a cylinder which is fitted with a ram which carries the cage and corves. The pressure of the water acting under the ram forces it up with the load it carries. The size of ram required depends upon the load and upon the head of water which is available.

Example.—A hydraulic ram is to be driven by a feeder of water 120 yards above the shaft bottom. If the total weight to be raised is 3 tons, find size of ram required, allowing a margin of 25 per cent. for friction.

The total weight to be raised is	6720 lbs.
Add $\frac{1}{4}$ for friction	1680 ,,
Total resistance to be overcome	8400 ,,

Head of water 120 yards = 360 feet

Pressure per square inch, $360 \times 0{\cdot}434 = 156{\cdot}24$ lbs.

The required area of ram equals the total resistance divided by the pressure per square inch $= \frac{8400}{156{\cdot}24} = 53{\cdot}7$ sq. inches.

The area of a circle = diameter$^2 \times 0{\cdot}7854$; $\therefore$ the diameter $= \sqrt{\frac{\text{area}}{0{\cdot}7854}}$, so that the diameter of the ram is $\sqrt{\frac{53{\cdot}7}{0{\cdot}7854}}$ $= 8\frac{1}{4}$ inches.

The principle upon which hydraulic jacks, or rams driven by artificial pressure, work is shown in Fig. 104. *a* and *b* are rams, *a* being much the smaller of the two. The cylinders in which the rams work are connected by the pipe *c*. If the area

of a is exactly 1 square inch, and a weight of 100 lbs. be placed upon it, the pressure per square inch at the bottom of a is exactly 100 lbs. As water transmits pressure equally in every direction, it follows that the pressure under b is also 100 lbs. per square inch. If the area of b is 50 square inches, the total pressure on b is $50 \times 100 = 5000$ lbs. This shows that the ratio of the total pressure on each ram varies with the respective areas, or, as the areas vary as the square of the diameters, the total pressure varies as the square of the diameters. By making the ram upon which the pressure is to be applied very small, and the ram through which the pressure is transmitted very large, enormous force can be obtained. It must be remembered that the gain in power is compensated by loss in speed. Thus the large ram in Fig. 104, having 50 times the area of the smaller ram, will travel at $\frac{1}{50}$ of the speed.

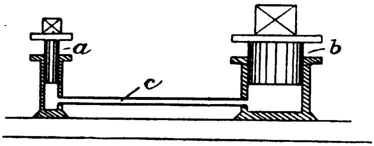


FIG. 104.—Hydraulic pressure.

In the ordinary hydraulic jack, the smaller ram is in the form of a single-acting ram-pump (see Chapter XXV.), and the pressure which is applied is multiplied by means of a lever.

Friction of Solids. Friction is the resistance encountered when one body slides upon another.

The laws governing the friction of solids are—

1. The resistance due to friction is proportional to the pressure upon the surfaces in contact.

2. The resistance due to friction is independent of the areas of the surfaces in contact.

3. The resistance is not greatly affected by the velocity at which one surface slides over the other.

4. The resistance is greatly affected by the nature and roughness of the surfaces in contact.

The truth of these laws can be demonstrated by the simple apparatus shown in Fig. 105. To prove the first law, sufficient

weight is hung on to the cord to cause the block to slide along the table; the total weight required to do this is the measure of the resistance due to friction. If another block of equal weight is placed on the top of the first block, the pressure between the surfaces is doubled, and it will be found that double the weight is required to pull the two along, thus proving that the resistance is proportional to the pressure.

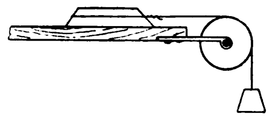


FIG. 105.—Friction of solids.

To prove the second law, the block is first slid along with the large surface in contact with the table, and then turned over on to the smaller side, exactly the same weight being required in either case. It will also be found that the weight required to drag the block along varies with the roughness of the surfaces in contact.

With surfaces of similar material the force of friction is a constant fraction of the pressure between the surfaces. This fraction is called the *coefficient of friction.* If the surfaces in contact are of iron and the block weighs 10 lbs., the force required to overcome the friction will be found to be about 2 lbs., or $\frac{1}{5}$ of the weight, so that the coefficient of friction of iron on iron is about $\frac{1}{5}$, or 0·2. The coefficients given by various authorities vary somewhat. The following are averages:—

Wood on wood or iron, dry ...	0·4 to 0·6
Iron on iron, dry	0·2
Axles, with ordinary lubrication	0·1

The amount of power absorbed by friction in bearings may be calculated as follows:—

A fly-wheel, weighing 12 tons, revolves in bearings 9 inches in diameter and 18 inches in length. What power is absorbed by friction when the revolutions per minute are 50? The circumference of the bearings is $9 \times 3·141 = 28·269$ inches, and as the revolutions per minute are 50, the distance moved

by the axle upon the bearings is $\frac{28{\cdot}269}{12} \times 50 = 117{\cdot}8$ feet per minute.

The pressure on the bearings is $12 \times 2240 = 26{,}880$ lbs., and taking the coefficient of friction at $\frac{{\cdot}1}{10}$, the total resistance is 2688 lbs., and the axle moves 117·8 feet per minute against this resistance, so that the foot-pounds of work done per minute are $117{\cdot}8 \times 2688 = 316{,}646$, which represents a horse-power of $\frac{316646}{33000} = 9{\cdot}59$ H.P.

If the length of the bearings is increased or reduced within reasonable limits, no alteration takes place as regards the friction, but if their diameter is increased, the power required to overcome the resistance of friction at a given number of revolutions per minute is proportionately increased, as the resistance has to be overcome over a greater distance. In practice the length of bearings is to a great extent governed by the question of lubrication.

Friction of Fluids.—The laws which govern the friction of fluids against solids, as, for example, water in pipes or air in airways, are—

1. The friction is independent of the pressure.
2. It varies as the area of the rubbing surface.
3. It varies as the square of the velocity.
4. It depends upon the nature of the rubbing surface.

The friction of fluids is fully dealt with in Chapter XX.

CHAPTER XVIII.

STEAM.

Heat.—All bodies are composed of minute particles called molecules. These molecules are held together by mutual attraction or cohesion, but are in a continual state of vibration. The hotter the body, the greater are the vibrations of the particles of which it is composed. In solid bodies the vibrations are limited in extent, but if enough heat is added to raise the vibrations sufficiently to overcome the cohesion of the particles and enable them to move about freely, the solid becomes a liquid. If still more heat is added, further separation of the molecules takes place, and the liquid becomes a gas. If the gas is enclosed, the pressure upon the vessel in which it is confined is due to the vibrating molecules beating against the sides, and is therefore greater as the temperature increases.

The temperature of a body is measured by thermometers and pyrometers (Chapter XXI.), and is the thermal state of the body in comparison with that of other bodies.

Unit of Heat.—The amount of heat necessary to raise the temperature of 1 lb. of water by 1 degree Fahr. is known as the unit of heat, or British thermal unit. Heat and work are convertible; a given amount of heat represents a given quantity of work, and *vice versâ*. In a steam-engine the heat given out by the burning of coal is converted into work; whilst, when a brake is applied to a revolving wheel, work is absorbed and heat is generated.

Joule found by experiment that 772 foot-pounds of work (Chapter XVII.) had to be expended in order to produce one unit of heat. This is known as Joule's mechanical equivalent,

which is, that 772 foot-pounds of work are the mechanical equivalent of one unit of heat.

Examples.—(1) To raise the temperature of 10 lbs. of water by 15 degrees Fahr. would require $10 \times 15 = 150$ units of heat, or the expenditure of $150 \times 772 = 115{,}800$ foot-pounds of work.

(2) An engine consumes 2 lbs. of coal per indicated horse-power per hour. What percentage of the heat given out is utilized?

Units of heat, 14,000 per pound of coal = 28,000
Equivalent foot-pounds of work, $28{,}000 \times 772$ = 21,616,000
Foot-pounds per minute utilized, 33,000; per hour, $33{,}000 \times 60$ = 1,980,000
Percentage of power applied which is utilized, $\frac{1980000 \times 100}{21616000}$ = 9·16 %

This example shows that with a high-class engine about $\frac{9}{10}$ of the heat in the coal is wasted. The loss is actually more than this, as the friction in engine and gearing has not been taken into account.

Specific Heat.—The specific heat of a body is the ratio of the quantity of heat required to raise the temperature of that body 1 degree to the quantity required to raise an equal weight of water 1 degree.

Water has the highest specific heat of any substance except hydrogen, the metals have the lowest. The specific heat of iron is 0·113; copper, 0·1; air, 0·238. So that it would take about nine times as much heat to raise the temperature of a pound of water 1 degree as it would to raise the temperature of a pound of iron by 1 degree.

Sensible and Latent Heat.—When heat which is applied to a substance raises the temperature of that substance, it is said to be sensible in the substance; but heat applied to a substance and not increasing its temperature is said to become latent.

If water is heated in an open vessel, it begins to boil when its temperature has reached 212 degrees Fahr. The temperature is

not increased by the application of further heat, but the water continues to boil, and becomes steam at the same temperature as the water. The heat required to change water at 212 degrees Fahr. to steam at the same temperature is known as latent heat, because its application does not result in an increase of temperature. The latent heat of a body is the quantity of heat which must be applied to it in order to change its form without raising its temperature. The latent heat of water is 966, so that 1 lb. of water at boiling-point requires the application of 966 units of heat to change it into steam, and 1 lb. of steam at 212 degrees Fahr. must be denuded of 966 units of heat to condense it into water at the same temperature.

Transfer of Heat.—Heat is transferred from one body to another by Radiation, Conduction, and Convection.

Radiation.—The transfer of heat in the form of rays is known as radiation; as, for example, the heating of the arch of a furnace by heat rays from the fire below it.

Conduction.—When one part of a body is heated, the heat gradually extends throughout the body by means of conduction; so that conduction is the passage of heat from one part of a body to another, or from one body to another body which is in contact with it. The heat from a boiler furnace passes through the plates to the water by conduction.

Some bodies conduct heat much more readily than others. Bodies that conduct heat freely are called conductors; those which conduct heat with difficulty are known as non-conductors. The metals are all good conductors; liquids, gases, and fibrous materials bad conductors.

Convection.—The passage of heat through liquids and gases by means of currents is known as convection. This is the most important manner in which liquids are heated. Convection currents always ascend, because the heated particles become less dense, and consequently rise. For this reason liquids must be heated from below: as the lower particles are heated they rise, and other cold particles take their place; these in their turn are heated, and rise, so that the whole of the liquid is exposed to the heat; whereas, if heated from near the top,

the water in the bottom of the vessel would remain there, and not come in contact with the heat.

Steam.—Steam is an elastic invisible gas generated by the heating of water. One cubic foot of water when changed into steam at atmospheric pressure occupies a space of 1642 cubic feet.

If 1 lb. of water at 32 degrees is heated, its temperature will be raised to 212 degrees by the application of about 180 units of heat. By the application of a further 966 units of heat, the water at 212 degrees becomes steam at the same temperature. If still more heat is applied, both the temperature and pressure of the steam rise in given proportions.

Steam in contact with the water from which it is generated (as is the case in the ordinary boilers) is said to be *saturated.* If steam is heated after generation until its temperature is higher than the temperature shown in the table given below, as corresponding to its pressure, it is known as *superheated steam.* Saturated steam, under a given pressure, has a fixed temperature and a given density; these temperature pressures and densities are shown in the following table:—

Absolute pressure of steam in lbs.	Temperature or boiling-point, in degrees Fahr.	Volume per lb. in cubic feet.	Total units of heat, in evaporating from 32 degrees Fahr.
1	102	330·36	1113·0
14·7	212	26·36	1146·6
20	228	19·72	1151·5
40	267	10·28	1163·4
60	292·6	7·01	1171·2
80	312·1	5·35	1177·1
100	327·7	4·33	1181·9
120	341·1	3·65	1186·0
140	352·9	3·16	1189·6
160	363·4	2·79	1192·8

This table shows that to generate steam at a pressure of 60 lbs. absolute (that is, 45 lbs. above atmospheric pressure), requires the application of 1171·2 units of heat, and that the

pressure could be raised to 120 lbs. by the application of only 14·8 additional units. This clearly shows the reason of the economy which results from the use of high-pressure steam, as much more work can be got out of it when used expansively than can be obtained from steam at a low pressure.

Expansion of Steam.—According to Boyle's Law (Chapter XX.), the pressure of a perfect gas at a constant temperature varies inversely as the space it occupies. *ab*, Fig. 106, is a cylinder in which the piston *c* moves through a stroke of

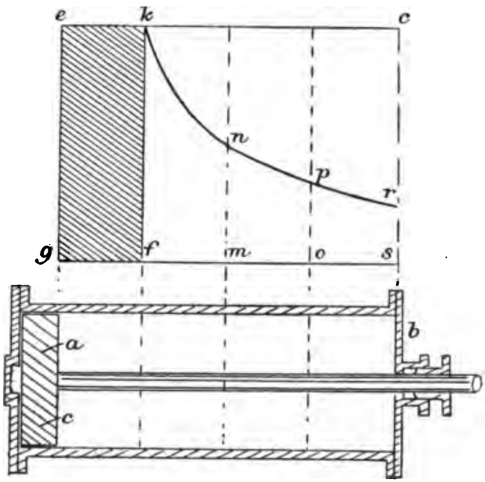


FIG. 106.—Expansion of steam.

20 inches. In the diagram drawn above the cylinder, the horizontal line *ec* represents the length of stroke, and the vertical line *eg* represents the pressure of the steam drawn to any convenient scale. If steam is admitted to the cylinder at a pressure of 100 lbs. per square inch, the line *eg* will represent 100 lbs. The pressure on the piston pushes it along, and, if the steam inlet to the cylinder is unrestricted, the pressure of 100 lbs. will be maintained during the whole length of the stroke. Suppose, however, that the steam valve

is closed when the piston has moved through 5 inches, or one quarter of its stroke, then the shaded area of the diagram represents the total volume of steam admitted to the cylinder, and the line *fk* represents 100 lbs., and is equal in length to *eg*. From this point the piston is moved solely by the expansive force of the steam. When the piston has moved through half the length of the stroke, the space occupied by the steam is doubled, and the pressure is halved, as by Boyle's Law, $pv = pv$ (p = pressure and v = volume); that is, $100 \times 1 = 2 \times$ pressure when expanded, and $\frac{100 \times 1}{2} = 50$. The line *mn* must be drawn half the length of *eg*, in order that it may represent the pressure of the steam when the piston has moved half its stroke. In the same way, when the piston has moved three-quarters of the length of its stroke, the steam occupies three times its original space, and its pressure is divided by three, so that the line *op*, which represents the pressure at $\frac{3}{4}$ stroke, is $\frac{1}{3}$ of the length of *ef*, and represents $33\frac{1}{3}$ lbs. Similarly, at the end of the stroke the steam has expanded four times, and *r*, which represents the pressure at that point, is $\frac{1}{4}$ the length of *eg*, and represents 25 lbs., which is the terminal pressure of the steam.

If a curve is drawn joining the points *k*, *n*, *p*, *r*, it will show the pressure of the steam at any point of the stroke. The steam consumed is represented by the shaded area *egkf*, but the work done is represented by the area *gers*; the unshaded portion being gained by the expansion of the steam.

In the example given above, the pressures given are absolute pressures; that is, they are measured from zero, and not from atmospheric pressure.

Steam Indicator.—The varying pressure of the steam in the cylinder of an engine is found by means of an "indicator."

Richards' indicator is shown in Fig. 107. It is screwed alternately into holes bored through the cylinder wall at either end. The steam pressing on the underside of the small

piston *a* raises it against the pressure of the spring *b*. Springs of varying strengths are used, to suit the different pressures of steam which are employed; thus a spring might have a deflection of 1 inch for 40 lbs.

The movements of the piston are transmitted by levers to the pencil *d*, the arrangement of the levers being such as to give the pencil a vertical movement. As the pressure in the cylinder increases or decreases, the small piston *a* rises or falls, and carries the pencil with it. *e* is a small brass drum, around which is wrapped the paper upon which the diagram is drawn by the pencil. This drum is connected by a cord to levers driven from the piston-rod; it does not revolve through the whole circumference of a circle, but moves through about $\frac{7}{8}$ of a revolution, and then back, being pulled one way by the cord, and back again by a spring. If steam is shut off from the indicator, the pencil is stationary, and draws a horizontal line upon the moving paper, and as the underside of the indicator piston is in communication with the atmosphere when not open to the steam cylinder, this line is an "atmospheric line;" but as soon as steam is admitted, the pencil moves up and down with the varying pressure, and its movement, together with the movement of the drum, results in a curved line being drawn upon the paper representing to scale the varying pressures of steam during the whole of the stroke.

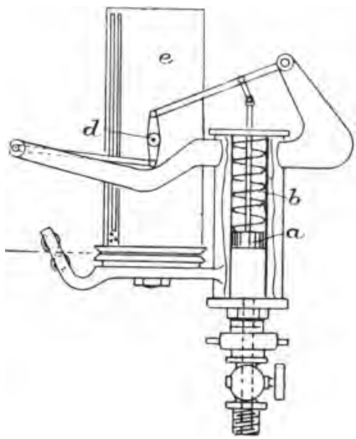


FIG. 107.—Richards' indicator.

Indicator Diagrams.—The sort of diagram given by a non-condensing engine is shown in Fig. 108. *a* is the commencement of the stroke, and *b* is the point at which the steam is

cut off; from *b* to *c* the piston is driven by the expansive force of the steam, and the sloping line shows the gradually decreasing pressure. At *c*, the end of the stroke is reached, and the exhaust valve is opened; from *c* to *d* the steam is pressing on the other side of the piston, whilst the steam at this side is flowing out through the exhaust. At *d*, the exhaust is closed, and steam is compressed behind the piston, forming a cushion in order to gradually arrest the momentum of the moving parts and reverse the direction of motion without shock. At *e*, the other end of the stroke is reached, and the steam valve reopened.

The shaded portion of the diagram is the "back pressure;"

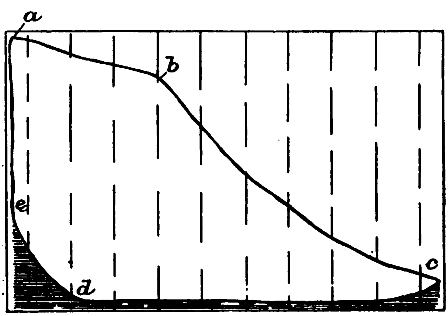


FIG. 108.—Engine diagram.

that is, pressure acting against the piston. High back pressures result in loss of power, and are caused by the exhaust outlets being too small or too long.

The average steam pressure is found by measuring the pressure on vertical lines drawn at equal distances along the length of the diagram; the pressures between the steam and back-pressure lines must be measured, not the actual pressures.

In Fig. 108, ten measurements are taken, as shown by the dotted lines; if these ten measurements are added and divided by 10, the average pressure is obtained. The lines must be measured with a scale corresponding to the pressure of the spring which is used in the indicator when the diagram is

taken; the scales of the springs commonly used are from 8 to 56 lbs. to an inch.

After the average pressure is obtained the indicated horsepower is calculated by the method given in Chapter XVII.

It will be noticed that the upper or steam line of the diagram, from *a* to *c*, Fig. 108, represents the steam pressure at one stroke of the engine, whilst the lower or exhaust line, *c* to *e*, represents the back pressure during the return stroke.

Steam-engines.—A simple non-condensing engine is an engine in which the steam does the whole of its work in one cylinder and exhausts direct into the atmosphere. There may

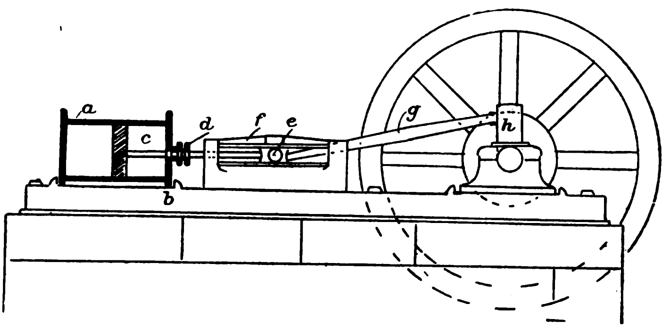


FIG. 109.—Horizontal non-condensing engine.

be one or two cylinders connected to one crank shaft by cranks placed at right angles to each other; each engine of a pair being complete in itself.

A simple engine is shown in Fig. 109. *a* is the cylinder, which is bolted to the bed-plate *b*; *c* is the piston-rod working through the stuffing-box *d*; *e* is the slide block, working between the slide bars *f*; *g* is the connecting-rod, and *h* the crank. The steam cylinder is provided with ports at either end; the opening of these ports is governed by a valve driven from an eccentric upon the main shaft. The port at one end of the cylinder is open to the steam, whilst the port at the other end is open to the exhaust.

Simple engines are not economical when the steam pressure is more than about 90 to 100 lbs. absolute, for the following reasons :—

1. A sufficient range of expansion cannot be obtained in one cylinder. In practice, steam cannot be expanded with efficiency more than about five times in simple engines.

2. When the cut-off is very early, the pressure upon the piston, and upon the crank pin, varies very greatly during the stroke; this leads to knocking and to unequal working. For example: If the initial pressure were 120 lbs. absolute, and steam were cut off at one-sixth of the stroke, the terminal pressure would be 20 lbs. (neglecting back pressure); so that at the commencement of the stroke the pressure per square inch would be 105 lbs. above the atmosphere, and at the end of the stroke only 5 lbs.

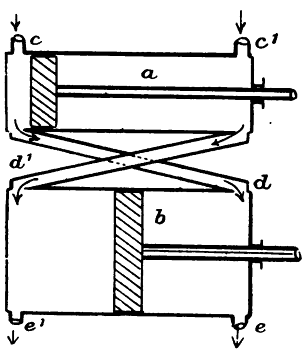


FIG. 110.—Compound engine.

3. When steam is expanded many times in one cylinder, the variations in temperature are excessive; this leads to loss of steam by initial condensation in the cylinder. In the example given above, the temperature of steam at 120 lbs. is 341 degrees, and of steam at 20 lbs. 228 degrees, or a difference of 113 degrees, so that steam at 341 degrees enters a cylinder cooled down by steam at 228 degrees.

Compound Engines.—The difficulties mentioned above with regard to working with high-pressure steam are overcome by the employment of compound engines. They consist of two steam cylinders, either set side by side, as shown in Fig. 110, or placed tandem fashion, one behind the other. Steam enters the high-pressure cylinder *a*, and does a certain amount of work; from *a* it exhausts into the low-pressure cylinder *b*, which is larger than *a*, where it does further work, and from which it exhausts into a condenser or into the atmosphere.

By this arrangement the expansion of the steam takes place in two cylinders, the second much larger than the first, so that a much greater range of expansion can be obtained; moreover, the pressure on the cranks is kept fairly uniform, and high ranges of pressure, and therefore of temperature, in one cylinder are avoided.

In Fig. 110 the steam enters the high-pressure cylinder at *c* and *c'*, and is expanded three or four times; it then exhausts into the low-pressure cylinder through the ports *d* and *d'*, where it is further expanded, and finally escapes into the atmosphere or into a condenser, through the exhaust ports *e* and *e'*.

When very high steam pressures are employed, three or four cylinders may be necessary.

Condensing Engines.—The pressure of the atmosphere averages nearly 15 lbs. per square inch, and as engines have to exhaust against this pressure, a considerable proportion of the steam pressure is wasted. By the use of condensers the effects of atmospheric pressure are removed from the back of the piston, and a greater rate of expansion is permissible. There are two kinds of condensers in general use—jet condensers and surface condensers.

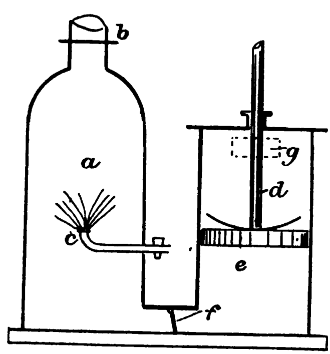


FIG. 111.—Jet condenser.

Jet Condensers.—In condensers of this class the steam is condensed by actual contact with cold water. Fig. 111 shows the form of condenser which is ordinarily employed in Cornish pumping engines; it is also employed in conjunction with other engines of both old and modern design.

a is the condenser into which the exhaust steam is admitted through the pipe *b*. This condenser is a cast-iron vessel, and is fitted with the injection nozzle *c*. *d* is the air-pump and *e*

the pump bucket, fitted with valves opening upwards. *f* is the foot valve set in the passage between condenser and air-pump, and opening towards the latter.

When the exhaust steam enters *a*, it meets the water from the injection nozzle and is condensed, causing a vacuum in the condenser.

The injection water and that which results from the condensed steam is drawn through the foot valve into the air-pump, at the upward stroke of the bucket, and delivered into the hot well through the port *g*.

Surface Condensers.—In these condensers the exhaust steam

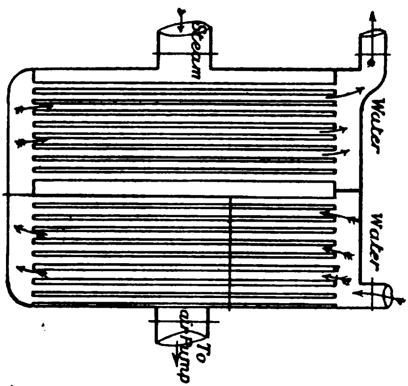


FIG. 112.—Surface condenser.

is condensed by coming in contact with the surface of pipes through which cold water is being continually pumped.

The advantage that surface condensers possess, is that the injection water does not mix with the condensed steam, so that the water which is used for steam generation may be pumped back into the boilers whilst hot, and used over and over again, and inferior water used for condensation.

A surface condenser is shown in outline in Fig. 112. The cold water is forced through the tubes by means of a circulating pump, and exhaust steam is brought from the engines into the space surrounding the tubes, and is there condensed by their

cold surface. The condensed steam is drawn off by an air-pump, filtered, and forced back into the boilers; and the water used for condensation is cooled in towers or in a pond, and used again when cool.

Some collieries have large central condensers, taking steam from all the engines; this is an advantage where engines work intermittently, as the supply of exhaust steam is regulated.

Boilers.—The pressure of the steam used at collieries has greatly increased during the last few years; at new collieries boilers are now seldom put down to work at less than from 100 to 120 lbs. per square inch, and in some cases these pressures have been greatly exceeded.

The Lancashire Boiler.—This type of boiler is almost universal at collieries. It is suitable for pressures up to about 160 lbs. per square inch, and is qualified to work with the bad water which is frequently found at collieries.

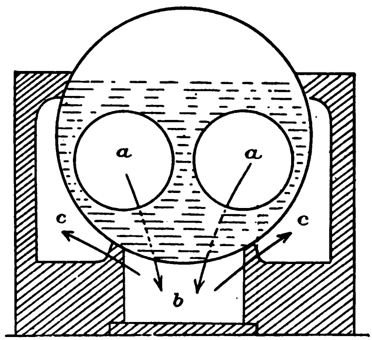


FIG. 113.—Cross-section through Lancashire boiler.

A good Lancashire boiler should evaporate from 7 to 9 lbs. of water per pound of coal which is consumed, and should consume about 20 lbs. of coal per square foot of firegrate area per hour. The Lancashire boiler consists of a plain cylindrical shell, with flat ends and two internal flues running the whole of its length. The usual size for heavy work is 8 feet in diameter by 30 feet in length. The ends are secured to the shell by being riveted to steel angles, and by gusset plates, and when the pressure is high, by tie bolts running right through the boiler.

Fig. 113 shows a cross-section through a Lancashire boiler and its seating, and Fig. 114 a longitudinal section. The

smoke and heat from the furnaces pass through the internal flues *a*, *a* to the back end of boiler, where the gases from both flues unite and pass back under the boiler through the bottom flue *b*, as shown by the arrows. On reaching the front the hot gases divide and pass to the back of the boiler through the side flues *c*, *c*, and from them go under dampers to the main flue and up the chimney. By this arrangement a large heating surface is secured, as the hot gases traverse the whole length of the boiler three times. The temperature of the flue gases on reaching the chimney should be between 400 and 600 degrees.

The manner in which the boiler is set is shown in the figures. The flues should be sufficiently large to enable them

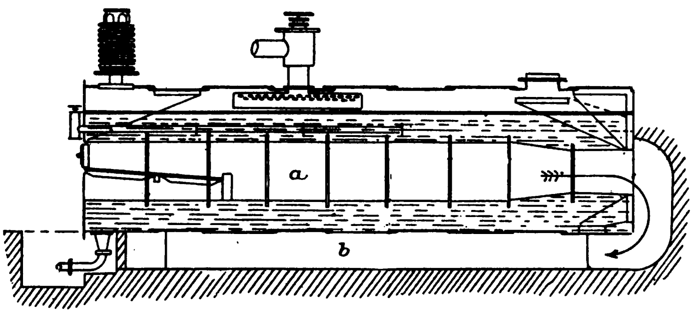


FIG. 114.—Longitudinal section through Lancashire boiler.

to be cleaned without trouble, and the area of brickwork in contact with the boiler should be small, as a large area harbours damp, and renders the plates which are covered inaccessible.

The following are the leading dimensions of a 30 feet by 8 feet Lancashire boiler, suitable for a working pressure of 100 lbs. per square inch.

Shell.—Made of nine belts, each belt of one plate $\frac{9}{16}$ inch thick; the ring joints lap riveted, and the longitudinal joints double riveted, and with internal and external cover plates.

Ends.—Each end in one solid plate, $\frac{5}{8}$ inch thick; ends secured to shell with steel angles and gusset stays, and strengthened with two longitudinal stays, each $1\frac{7}{8}$ inch in diameter.

Flues.—Each flue in ten rings, $\frac{7}{16}$ inch thick, 3 feet 3 inches in diameter at the front, and 2 feet 10 inches in diameter at the back.

Fittings.—One steam stop valve, 6 inches in diameter; one check feed valve, $2\frac{1}{2}$ inches in diameter. One manhole, 16 inches in diameter; one mudhole, 15 inches by 12 inches; two dead-weight safety valves, 3 inches in diameter; two water-gauges; one 10-inch pressure gauge; one blow-off tap, $2\frac{1}{2}$ inches in diameter; one fusible plug in each flue.

The shell, ends, and flues of mild steel having a tensile strength of not more than 30, or less than 26 tons per square inch.

A boiler such as the above would have a firegrate area of about 38 sq. feet, and an effective heating surface of about 900 sq. feet. When consuming 20 lbs. of coal per square foot of firegrate area per hour, the coal consumption would amount to 760 lbs., and the water evaporated per hour would be about 6000 lbs. Colliery engines consume from 20 to 60 lbs. of steam per indicated horse-power per hour, according to the type and condition of the engines; so that a Lancashire boiler, 30 feet by 8 feet, should supply sufficient steam to supply engines developing from 100 to 300 indicated horse-power.

Water-tube Boilers.—These boilers are not as yet very largely used about collieries, but the increased pressure of steam which is now employed is likely to lead to an extended application of boilers of this type.

The advantages they possess over Lancashire boilers are that they can be made to work at higher pressures, evaporate more water per pound of coal consumed, and take up less floor space. On the other hand, they are more complicated, not so easily repaired, and, for satisfactory working, good water is essential, or the tubes get stopped up and are burned.

Water-tube boilers consist of a series of tubes through which water is made to circulate, and around which the hot gases from the furnace are led.

The tubes are connected to one or more cylindrical iron drums, which serve as a reservoir for the hot water and steam.

Fig. 115 is a section through a Stirling boiler.

There are three top drums, *a*, *b*, and *c*, connected by tubes to the bottom drums *d* and *e*; the drums are from 3 feet to $3\frac{1}{2}$ feet in diameter, and the outside diameter of the tubes is $3\frac{1}{4}$ inches.

About $\frac{1}{3}$ of the capacity of the top drums is occupied by water and the remainder by steam; the lower drums as well as the whole of the tubes are, of course, filled with water. Owing to the large surface area of the tubes, the heating surface is very great, and as the water circulates freely, a large quantity can be evaporated.

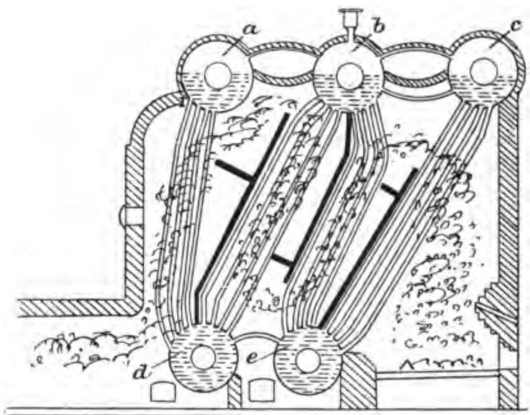


FIG. 115.--Section through Stirling boiler.

The tubes are all nearly vertical, so that sediment does not settle in them, but falls into the two lower drums, from which it can be cleared periodically by blow-off taps. The length of the drums varies from 4 feet 8 inches in the smallest size up to 18 feet in the largest.

Economizers.—The gases from the boiler furnaces enter the main flue at a high temperature, and, if they are allowed to escape directly into the chimney, a considerable amount of heat is wasted.

This waste heat may be utilized by "fuel economizers."

Green's economizers consist of a stack of cast-iron pipes placed vertically in the main flue from the boilers, and through which the feed water is pumped on its way to the boilers. The tubes are 9 feet in length and about 4½ inches in external diameter; they are connected by boxes at the top and bottom. The feed water is introduced at the end of the apparatus nearest the chimney, passes through the tubes, where it is heated to a high temperature, and into the boilers from the other end.

The pipes are fitted with movable scrapers, which are driven by a small engine, and made to travel slowly up and down the pipes in order to prevent the accumulation of soot, which, being a bad conductor of heat, would destroy the efficiency.

By the use of economizers the temperature of the flue gases is lowered by 200 to 300 degrees, and the temperature of the feed water is raised to about 250 degrees.

Colliery Consumption.—The amount of fuel consumed at a colliery is usually expressed as a percentage of the output. This percentage varies from about 2½ at new collieries, to as much as 15 or 16 in the case of old places fitted with uneconomical engines, and having much water to pump. At some collieries the whole of the steam that is required is raised by the heat given off from coke ovens.

Compressed Air.—The prime source of the power used at collieries is in almost every case steam, but it frequently happens that power is required at points far distant from the boilers where the steam is generated. Pumps, coal-cutting machines, or hauling machines may be situated a mile or two away from the boilers, and special methods must be adopted for transmitting power to them. The two methods of transmitting power which are universally employed are electricity and compressed air. In transmitting power by compressed air, the air is compressed at the surface, and conveyed at a high pressure, in pipes, to the engines, in which it does work by expanding again to atmospheric pressure.

The application of compressed air is similar to the

application of steam, except that the latter is condensable and the former is not.

The physical properties of air which affect the question of air compressors are—

1. The pressure of air varies inversely as its volume, provided the temperature remains the same.

2. The pressure varies as the absolute temperature (see Chapter XX.) if the volume is kept constant; or, if free to expand and under a constant pressure, the volume varies as the absolute temperature. The pressure of air may be measured either in pounds per square inch or in atmospheres,

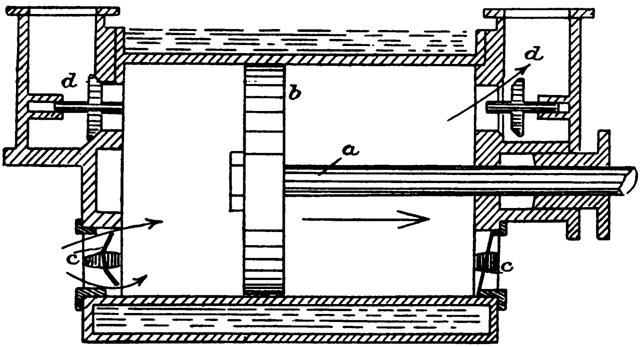


FIG. 116.—Section through air-compressing cylinder.

one atmosphere equalling 14·7 lbs. per square inch. It must be remembered that atmospheric, or free air, has a pressure of 1 atmosphere before being subjected to any pressure. The pressures shown on a steam gauge are measured from atmospheric pressure, and to reduce gauge pressures to atmospheric pressures, 14·7, or roughly 15 lbs., must be added to the former. For example, if the pressure on a gauge reads 60 lbs., the absolute pressure is 75 lbs.

A section through an air-compressing cylinder of ordinary type is shown in Fig. 116. The air-cylinder is usually placed behind and in line with the steam cylinder, the back piston-rod of the engine being coupled to the piston-rod of the air-

compressor. In Fig. 116, *a* is the piston-rod, *b* the piston, *c* the air inlet valves opening inwards, and *d* the outlet or delivery valves opening outwards, and communicating with the air-receiver. The cylinder is surrounded by a water jacket, or box through which water constantly circulates. As the piston moves in the direction of the arrow, air at atmospheric pressure is drawn into the cylinder behind it, through the inlet valves; at the same time the air in front of the piston is being compressed, and is forced through the outlet valve as soon as its pressure exceeds that of the air in the receiver. At the return stroke, air is drawn into the cylinder through the inlet valves at the other end, whilst the air drawn in at the previous stroke is being compressed.

The behaviour of air during compression will be understood by an examination of the diagram given in Fig. 117.

ab is the air-cylinder fitted with valves and piston, as shown in Fig. 116. Taking the piston to be at the end of the cylinder marked *a*, and travelling slowly in the direction of the arrow, the whole space in front of the piston is occupied with air at atmospheric pressure, or, say, 14·7 lbs. per square inch absolute. By the time that the piston has moved through $\frac{1}{10}$ of the length of the cylinder, the volume of air is reduced to $\frac{9}{10}$ of its original bulk, and, according to Boyle's Law, its pressure is increased in inverse proportion, and has become $\frac{10}{9}$ of 14·7 lbs., that is 16·33 lbs.

Similarly, when the piston has moved through $\frac{2}{10}$ of the stroke, the volume is reduced to $\frac{8}{10}$ and the pressure increased to $\frac{10}{8}$, so that the pressure at that point is $14·7 \times \frac{10}{8} = 18·37$ lbs. In like manner, the pressure of the air at any point in the stroke can be determined. The line marked "isothermal curve" in the diagram shows the pressures generated during compression plotted to a scale.

The pressure in the cylinder continues to rise until it is equal to the pressure in the receiver. As soon as this point is reached, the piston forces the compressed air through the outlet valve. For example, if the pressure in the receiver were 4 atmospheres, or 58·8 lbs. absolute, air would be

delivered when the piston had passed through $\frac{3}{4}$ of its stroke, because the pressure of the free air would be increased fourfold by the time that its volume was diminished to $\frac{1}{4}$.

It is important to remember that the pressure varies inversely as the volume, *provided only that the temperature remains unchanged*, and the isothermal curve and figures given above are based on the assumption that this is so. In practice, however, a large increase in temperature takes place, unless the air-cylinder is very efficiently cooled. Whenever work is done, heat or its equivalent is expended, and this heat is accompanied by a large increase in volume or pressure. The "isothermal" curve is the curve that would result if air were compressed at a constant temperature; and the "adiabatic" curve drawn above it shows the result of compressing air without any cooling arrangements; the difference between

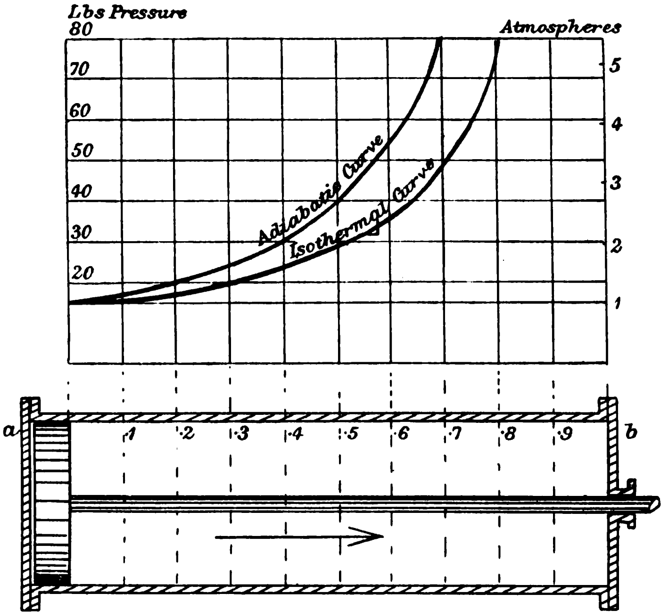


FIG. 117.—Diagram showing pressure of air during compression.

the two curves being the result of the heating of the air during compression.

The adiabatic curve is of course always the higher; thus at 0·5 of the stroke the pressure due to isothermal compression is 2 atmospheres, or 29·4 lbs., whilst with adiabatic compression the pressure is 39 lbs.; at 0·7 of the stroke the pressures are 49 and 81 lbs. respectively.

As the curves represent the pressure of the air behind the piston, they must also represent the pressure required to drive the piston along, so that they show that much more pressure, and therefore more power, is required with adiabatic than with isothermal compression.

This excess of power is caused by the heating of the air; and as the air cools before it is used, it follows that this extra power is wasted. For example: whilst compressing air, 2 cubic feet may expand by the heat to 3 cubic feet, and sufficient power has to be used to compress those 3 cubic feet, but when the air comes to be used, it has cooled and shrunk again to 2 cubic feet, and $\frac{1}{8}$ of the power is wasted.

In order to avoid this loss, the air-cylinder is kept as cool as possible by a water-jacket, and in some cases water is injected into the cylinders during compression.

The curve obtained in practice is always between the adiabatic and isothermal curves. The more perfect the cooling arrangements, the more nearly it approaches the latter.

The temperatures generated by compressing air are shown in the following table:—

Pounds per square inch absolute.	Atmospheres.	Temperature in degrees Fahrenheit.
14·7	1	60·0
29·4	2	175·8
44·1	3	255·1
58·8	4	317·4
73·5	5	369·4
88·2	6	414·5
102·9	7	454·5
117·6	8	490·6

These high temperatures act adversely in other directions; they render proper lubrication of the cylinders very difficult, and when the atmospheric air is drawn into a hot cylinder it expands, so that the weight of air which is actually taken into the cylinder at each stroke is diminished.

The quantity of air delivered by a compressor at a given pressure may be calculated as follows:—

How many cubic feet of air at 60 lbs. pressure will an air-compressor, having an air-cylinder 30 inches in diameter by 5 feet stroke, deliver when making 40 revolutions per minute? The pressure on the gauge being 60 lbs., the absolute pressure is 75 lbs. (Atmospheric pressure is usually taken at 15 lbs. per square inch in practice.)

Area of cylinder $2{\cdot}5^2 \times 0{\cdot}7854 = 4{\cdot}91$ sq. feet.
Feet per minute travelled by piston $40 \times 5 \times 2 = 400$ feet.
Volume of free air taken into cylinder per minute $4{\cdot}91 \times 400 = 1964$ cub. feet.
Pressure of free air = 15 lbs.
Pressure of compressed air = 75 lbs.
Ratio of pressure = 15 to 75 lbs.
" volumes = 75 to 15 lbs.

$$\text{Actual volume } \frac{1964 \times 15}{75} = 392{\cdot}8 \text{ cub. feet.}$$

This rule may be stated as follows:—

Add 15 to the pressure on gauge, and divide the sum by 15, which gives the number of atmospheres; divide the cubic feet of free air by this, and the quotient is the cubic feet of compressed air.

The volume of air found in this manner is the theoretical quantity delivered by isothermal compression. A deduction of about 25 per cent. should be made for the losses by heating, clearance, and leakage through valve and pistons, etc.

The pressure on the steam piston is greatest during the commencement of the stroke, and after the steam is cut off the pressure rapidly decreases; whereas the pressure in the air-cylinder is zero at the commencement of the stroke, and rapidly

increases until it reaches the maximum, and air is forced through the delivery valve. From this it follows that when both steam and air pistons are attached to one rod, the pressure of steam is at its minimum when the pressure of air is at its maximum, and *vice versâ*. This difficulty is overcome by constructing air-compressors in pairs, with cranks coupled at right angles; and by the provision of a heavy fly-wheel. The excess of power is absorbed by the fly-wheel during the earlier part of each stroke, and given out during the latter part.

Air-compressors work most efficiently when the pressure is low, but the engines they drive are more efficient with high pressures.

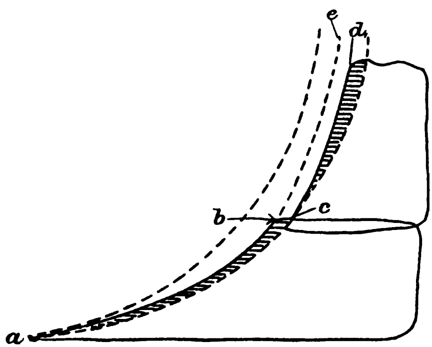


FIG. 118.—Air-compressing in stages.

The most economical method of transmitting power by compressed air is to use a high pressure, and compress the air in stages. Two air-cylinders are employed, the low pressure and high pressure, the former being the larger of the two. Atmospheric air is taken into the low-pressure cylinder and compressed to about 30 to 40 lbs.; it then passes through an inter-cooler, where it is cooled by coming in contact with the surface of pipes through which cold water is forced. From the inter-cooler the compressed air passes into the high-pressure cylinder, and its pressure is raised by further compression to about 100 lbs.

Fig. 118 shows the effects of compressing air in stages.

The part of the curve *ab* shows the pressure in the low-pressure cylinder; the effect of the inter-cooler is shown by the decreased volume at *bc;* the upper part of the diagram, *cd*, is the line of pressure in the high-pressure cylinder. The shaded portion represents waste work, owing to the impossibility of keeping the temperature constant by the water-jackets. If the air had been compressed in one cylinder, the pressure curve would have been about that of the line *abe*, so the saving effected by compressing by stages is represented by the spaces *e*, *b*, *c*, *d*.

Air Mains.—For efficient working, the velocity of the air through the mains should not exceed 50 feet per second, and large receivers should be employed, placed as near to the air-motors as possible.

Compressed air is very safe and convenient, but it usually gives a very low efficiency. This is not so much due to the system itself as to the manner in which it is employed, the machinery being generally of an uneconomical type. The efficiency of compressed-air plants may be improved by the use of reheaters, in which the compressed air is heated before it enters the cylinders of the engines in which it is to be used.

In order to combine the economy of transmission of power by electricity with the convenience and safety of compressed air, electrically driven air-compressors have been designed.

By fixing these in the intakes the risk of firing gas is reduced to a minimum, and long air mains are avoided.

CHAPTER XIX.

GASES.

ALL substances in the earth or atmosphere are built up of certain *elements;* these elements may be in their simple state, may be mixed together, or may be chemically combined. An element cannot by any known method be split up into other and simpler substances, neither can it be destroyed.

There are about seventy known elements, among which the following are the most important:—

Calcium.	Iron.	Silica.
Carbon.	Lead.	Sodium.
Chlorine.	Nitrogen.	Sulphur.
Copper.	Oxygen.	Tin.
Gold.	Phosphorus.	Zinc.
Hydrogen.	Silicon.	

A Compound is a body composed of two or more elements which have entered into chemical combination. The properties of a compound may be entirely different from the properties of the elements of which it is composed, and it cannot be split up into its elements except by chemical means. Water is an example of a chemical compound being a combination of oxygen and hydrogen.

When two or more elements unite in such a manner as to be capable of being split up by mechanical means, they form a *mixture.* Air is an example of a mixture; it is composed of oxygen and nitrogen. They are not chemically combined, but merely *mixed together.*

A compound or mixture is composed of elements; elements are made up of *molecules*, and molecules of *atoms*.

An atom is the smallest particle of matter capable of entering into a chemical combination.

A molecule is the smallest particle of matter which can exist in a free state.

The weight of an atom is not known, but the relative weights of atoms are known.

The atomic weight of an element is the weight of an atom of that element as compared with an atom of hydrogen. The atomic weights represent the proportions by weight in which the various elements combine with each other, and no element ever combines with another element except in proportion to its atomic weight or to some multiple of its atomic weight.

The Atmosphere.—The atmosphere, or air, is a mechanical mixture of nitrogen and oxygen; it also contains small and variable quantities of carbonic acid, aqueous vapour, and ammonia. The composition of air, when pure, is as follows:—

	By weight.	By volume.
Nitrogen	77	79
Oxygen	23	21

Oxygen.—Chemical symbol, O. Specific gravity (air being 1), 1·105. Oxygen is the support of animal life. All animals must breathe it in its uncombined state, or die from oxygen starvation. Oxygen combines readily with many substances, and when the combination is rapid, and is accompanied by heat and flame, it is known as *combustion.* All substances which burn in air burn more freely in pure oxygen; and many substances which will not burn in air burn quite readily when immersed in oxygen. Oxygen is used to revive men who have been partially suffocated by poisonous gas, or when, owing to illness, vitality becomes very low; but, if breathed for long in its pure state, it would cause death, owing to the too rapid action of the heart and other functions.

Nitrogen.—N. Sp. gr. 0·971. Nitrogen will not support life or combustion, but it serves to dilute oxygen and render it fit to be breathed. It is not poisonous, but, as it will not support life, any one breathing it in its pure state would die for want of oxygen.

When a man or other animal breathes, he inhales pure air, but exhales a mixture of unchanged air, free nitrogen and carbonic acid gas. Plants, on the other hand, absorb carbonic acid gas and give out oxygen, but retain the carbon. When at rest, a man inhales about 550 cubic inches of air per minute, and the mixture exhaled contains about 4 per cent. of carbonic acid gas, so that each man produces about 22 cubic inches of carbonic acid gas per minute. When a man is at work he breathes much more rapidly, and consequently produces much more carbonic acid gas.

The air in mines is vitiated by the following causes :—

1. Presence of noxious gases given off by the strata.
2. The breathing of men and horses.
3. The burning of lights, and
4. The firing of explosives.
5. The admixture of coal and other dust.
6. The absorption of oxygen by coal.
7. In some cases the presence of gob-fires.

The first is by far the most important cause of vitiation, and, in some mines, much air may be required to dilute the fumes given off from the firing of explosives.

Gases found in Mines.—The following are the "noxious" gases found in mines :—

	Symbol.	Sp. gr. (air being 1).
1. Carburetted hydrogen, methyl hydride, marsh gas, fire-damp, or "gas" ...	CH_4	0·559
2. Carbon dioxide, carbonic acid gas, black damp, or "damp"	CO_2	1·529
3. Carbon monoxide, or carbonic oxide, or "white damp"	CO	0·975
4. Sulphuretted hydrogen, or hydrogen sulphide	H_2S	1·175

Carburetted Hydrogen Gas.—This gas is given off naturally in most coal mines, though there are some mines, and even whole coal-fields, in which it is never met with. It results from the decay of vegetable matter, and may exist in the strata at extremely high pressures. The pressure has been ascertained in several collieries by boring holes into the coal and plugging them tightly up, leaving a tube through the plug, to which a pressure gauge could be attached. In some cases a pressure of over 400 lbs. per square inch was registered, although the volume of gas given off was inconsiderable.

Gas may be given off from the coal face in the form of minute sprays, which escape from the pores of the coal; or it may be given off from *blowers*, which may continue to produce large volumes of gas for years. Some seams are liable to sudden outbursts of gas, in which a huge volume of gas is given off, but gradually decreases and dies away in a few days. Gas is not necessarily given off from the coal itself, but often from the adjacent strata; a thin seam of coal, above or below the one being worked, is often productive of much gas. Goaves often contain a large quantity of gas, some of which escapes into the roadways and workings when the pressure upon it is reduced.

Carburetted hydrogen alone cannot be breathed, but, when mixed with air, has no effect on man until it forms about 50 per cent. of the mixture. Fatal accidents occasionally occur through men going into an accumulation of almost pure gas; they are apt to devote the whole of their attention to its explosive properties and forget that it cannot be breathed.

Carburetted hydrogen alone is not explosive, but only when mixed with air in certain proportions. When mixed with about five times its bulk of air it explodes feebly. The most explosive mixture is reached when mixed with eight to ten times its bulk of air, and when mixed with more than fifteen volumes of air it ceases to be explosive.

Many experiments have been made to ascertain the exact proportions of air and fire-damp which are explosive; all differ slightly. This is probably because the composition of the gas

used in the various experiments was not the same. Gas as given off in mines usually contains impurities.

Carburetted hydrogen, being so much lighter than air, is always found near the roof and in the highest places in the mine. For this reason dip roads are always more easily ventilated than roads driven to the rise. Owing to that property known as *diffusion*, which all gases possess, carburetted hydrogen and air, when in contact, do not form distinct layers—as, for example, is the case of oil and water—but gradually mingle with each other, and pass imperceptibly from pure CH_4 at the highest points to pure air at the lowest.

The presence of this gas is detected by noting the behaviour of the flame of a safety-lamp. When about 2½ per cent. of gas is present, the flame flickers, and is slightly "drawn;" as the percentage of gas increases, a slight blue cap is formed, which becomes more marked until about 6 per cent. of gas is present, when the gas burns in the lamp.

Carbonic Acid Gas.—This gas is given off naturally from the strata in some mines, and is also produced by the breathing of men and animals, and by the burning of lights and explosives. It is usually much more prevalent in shallow than in deep mines, and, being much heavier than air, it accumulates near the floor and at the bottom of sumps or wells. Carbonic acid gas results from the combustion of carbon in a plentiful supply of oxygen. When 15 per cent. of this gas is present in air, lights are extinguished, and the mixture becomes fatal to life when 25 per cent. is present. The "black damp," or "choke damp," found in mines or wells, is often a mixture of nitrogen and carbonic acid gas. When the barometer rises, it is a sign that the pressure of the atmosphere is increased (Chapter XXI.), and air is forced into the strata, undergoing a process of oxidation whereby the oxygen is absorbed, leaving only nitrogen; when the pressure is decreased, this nitrogen issues from the strata, carrying with it a small percentage of carbonic acid gas. Dr. Haldane has clearly shown that the choke damp found in wells is poisonous, not so much from the amount of CO_2 that is present, but from the

absence of oxygen. The presence of carbonic acid gas or black damp is detected by the light of a candle or lamp being dimmed and finally extinguished. Great care should be exercised in travelling a road containing damp. It is usually easier to go forward than to return, as in going forward the gas which lies near the floor is stirred up, making the return journey much more difficult.

Carbonic Oxide or Carbon Monoxide.—This gas is not given off naturally in mines, but results from incomplete combustion. Whenever carbon is burned in an insufficient supply of oxygen, carbon monoxide is formed. It may result from a gob-fire, from blasting when the explosive is bad, and from an explosion of gas and coal-dust. It is an extremely poisonous gas, and when 0·5 per cent. is present in the air it will cause death, if breathed for a prolonged period, and when 1 per cent. is present, death results in a very short time. The first symptom of carbonic oxide poisoning is dizziness, palpitation, and shortness of breath; finally the legs give way, and the patient becomes unconscious and dies. Dr. Haldane, who has investigated the subject very thoroughly, suggests that a live mouse should be taken into the mine after an explosion has occurred. A mouse suffers from the same symptoms as a man, but the effect is much more rapid. The pressure of carbon monoxide can be detected by the effect that it has on a mixture of blood and water. If the solution contains about 1 per cent. of normal blood, it is of a pale yellow colour, but the colour changes to pink if shaken up in air containing carbon monoxide.

The best remedy for carbon monoxide poisoning is the administration of oxygen. Artificial respiration should be applied if necessary, and the patient should be kept warm and quiet.

Sulphuretted Hydrogen.—This gas is generated by the putrefaction of animal and vegetable matter, and by the oxidation of sulphates. It is also produced by the decomposition of iron pyrites, and in some cases by the explosion of blasting powder. It is sometimes found in old water-levels and in old workings which have been filled with water. It is a highly

poisonous gas; 0·1 per cent. is said to cause death if breathed for any length of time. This gas is readily recognized by its unpleasant smell.

After-damp.—The mixture of gases which results from an explosion of gas, or of gas and coal-dust, is known as after-damp.

The composition of after-damp depends upon the proportions of gas, air, and coal-dust which take part in the explosion. At one time it was thought that the atmosphere of a mine, after an explosion had taken place, consisted at first of nitrogen, carbonic acid gas, and steam, and, after the steam had condensed, of nitrogen and carbonic acid gas. That this is not so is proved by the fact that fires caused by the explosion, and even men's lamps, have continued to burn in places in which all the men have been killed.

According to Dr. Haldane, the chief cause of death in colliery explosions is the presence of carbon monoxide. He found that in three large explosions 77 per cent. of the men killed were not killed by the force of the blast, but by the after-damp, and their bodies showed every symptom of death by carbon monoxide poisoning.

CHAPTER XX.

VENTILATION.

In common with other gases, air has the following properties: It is elastic; it has weight; and it has the property of inertia, which means that it never moves without the application of force, and, if once set in motion, never stops unless exposed to some resistance—the resistance usually being friction.

In order to produce the currents of air for the ventilation of mines, advantage is taken of the facts which are expressed in what are known as Charles' and Boyle's Laws.

Charles's Law.—This law states that the volume of a gas varies directly as the absolute temperature, when the pressure is constant.

The "absolute zero" is -459 degrees Fahr., or -273 degrees Cent., and is supposed to be the lowest temperature which is attainable. The absolute temperature, using Fahrenheit's scale, is the temperature given by the thermometer, $+459$. Thus the absolute temperature of a gas at 50 degrees Fahr. is $50 + 459 = 509$ degrees; and of a gas at 100 degrees Fahr., $100 + 459 = 559$ degrees. Thus, if 100 cub. feet of air at 50 degrees Fahr. were heated to 100 degrees, the relative volumes would be in the proportions of 509 to 559, and the actual volume after the increase in temperature $\frac{559 \times 100}{509} =$ 109·82 cub. feet.

The *total* weight of air would of course remain unchanged, so that the weight *per cubic foot* would be decreased by about one-eleventh. Hence it follows from this law that the weight per cubic foot of air decreases when the air is heated.

Boyle's Law.—This law states that the volume of a gas varies inversely as the pressure, if the temperature remains constant. Thus if the pressure on a body of gas is doubled its volume is halved. Although the air is compressed to half its former bulk, the total weight of air is unchanged, so that the weight per cubic foot is doubled; from this it follows that the weight per cubic foot of a gas varies directly as the pressure upon it. The pressure upon free air is that due to the weight of the atmosphere, and is usually expressed in inches of mercury (see Chapter XXI.). The alteration in volume due to the alteration in pressure is calculated as follows: Find the alteration in volume of 50 cub. feet of air if the barometer falls from 30·74 to 28·16.

As 28·16 : 30·74 : : 50 to altered bulk.

So that the altered bulk is $\frac{30{\cdot}74 \times 50}{28{\cdot}16} = 54{\cdot}58$ cub. feet.

These two laws show that the weight of a given volume of air depends upon the temperature and pressure, and as these vary constantly, the weight of air also varies from time to time. It has been calculated that 459 cub. feet of air weigh 1·3253 lbs. when the temperature is 0 degrees Fahr., and the pressure is equal to 1 inch of mercury. The weight of 1 cub. foot of air under these conditions is $\frac{1{\cdot}3253}{459}$ lbs., and the weight at any pressure is $\frac{B \times 1{\cdot}3253}{459}$, where B is height of barometer in inches, and, as the weight varies inversely as the absolute temperature, the weight at any pressure and temperature is found by the formula $W = \frac{B \times 1{\cdot}3253}{459 + t}$. For example, find the weight of a cubic foot of air when the barometer reads 30·6 inches and the thermometer 65 degrees.

$$W = \frac{30{\cdot}6 \times 1{\cdot}3253}{459 + 65} = 0{\cdot}07739 \text{ lb.}$$

Pressure producing Ventilation.—Mines are ventilated entirely by gravity; for in order to produce a current of air there must

be two columns of air of different densities, that is, the weight per cubic foot of air in the one column must be different to the weight per cubic foot in the other. The two columns are those found in the upcast and downcast shafts, or their equivalents; the difference in density is caused either by a furnace, which heats the air in the upcast and so reduces its density (Charles's Law); or by a fan, which reduces the pressure below that of the atmosphere (Boyle's Law).

Fig. 119 illustrates the manner in which the pressure giving rise to a ventilating current is calculated. *a* and *b* are two shafts, each 1000 feet in depth: the air in *a* weighs 0·08 lb. per cubic foot, and in *b* 0·06 lb. per cubic foot. This difference in weight may have been brought about either by a fan or furnace, as explained above. If the communication between the bottoms of the two shafts is closed by a stopping, the pressure per square foot on either side of it is equal to the weight of a column of air 1 sq. foot in area in and above each of the shafts; but the columns of air above the tops of the two shafts are of the same density, so that they exactly balance each other, and can be omitted from the calculation.

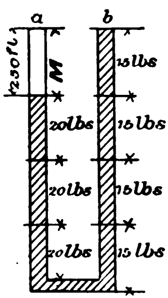


FIG. 119.—Motive column.

Ignoring the columns of air above the shaft tops, the pressure per square foot on the side of the stopping communicating with *a* is equal to the weight of 1 cub. foot of air multiplied by the height of the column, which is $0·08 \times 1000 = 80$ lbs., and the pressure per square foot on the other side is $0·06 \times 1000 = 60$ lbs. The difference in pressure is 20 lbs. per square foot, which is the pressure producing ventilation. If a water-gauge (Chapter XXI.) is placed on the door, the water in the leg communicating with *a* will be depressed, and as 5·2 lbs. pressure per square foot balances 1 inch of water, the depression will amount to $\frac{20}{5·2} = 3·85$ inches. As the pressure is greater in *a* than in *b*, *a* will be the downcast and *b* the upcast shaft.

Motive Column.—It will be seen from the above that the pressure producing ventilation may be expressed either in pounds per square foot or in inches of water-gauge; but it may also be expressed in feet of motive column. The motive column is a column of air 1 sq. foot in area, and of the same weight per cubic foot as the air in the downcast shaft, and of such a height that its weight is equal to the ventilating pressure. In the example given above, the ventilating pressure is 20 lbs. per square foot, and the weight of air in the downcast shaft is 0·08 lb. per cubic foot: hence the length of the motive column is $\frac{20}{0·08} = 250$ feet. Looking at the question in a slightly different light, we see that the weight of the air column in *b* is 15 lbs. for every 250 feet, and in *a* it is 20 lbs. for every 250 feet, so that a column in *a* 750 feet long balances a column in *b* 1000 feet long. This leaves a column of air in *a* 250 feet long, which is unbalanced and free to give rise to the motion of the air; this is known as the motive column. The velocity at which the air would circulate if the stopping were removed, and no friction existed, would be equal to the velocity that a body would acquire in falling through a height equal to the length of the motive column. In practice, however, most of the ventilating pressure is required to overcome friction, only a very small proportion being spent in giving rise to velocity.

The relation between pressure, water-gauge, and motive column is expressed in the following formulæ,
where P = pressure in pounds per square foot,
WG = water-gauge in inches,
M = motive column in feet:—

$$M = \frac{P}{\text{average weight of a cubic foot of air in downcast}}$$

$$M \times \text{average weight of a cubic foot of air in downcast} = P$$

$$WG = \frac{P}{5·2} \qquad P = WG \times 5·2$$

The Friction of Air in Mines.—As the air traverses the

roadways of a mine it rubs against the roof, floor, and sides, and this gives rise to friction.

The laws governing the friction of air in mines are as follows:—

The pressure required to overcome the resistance caused by friction varies—

(*a*) Directly as the rubbing surface;

(*b*) Inversely as the area of the airway;

(*c*) Directly as the square of the velocity;

(*d*) According to the nature of the rubbing surface, being greater with rough than smooth surfaces.

The Rubbing Surface.—Seeing that the friction is caused solely by the air coming in contact with the "rubbing surface," it follows that the greater the rubbing surface the greater the friction, other things being equal. The rubbing surface is the total area exposed to the air, and is obtained by multiplying the perimeter of the airway by its length. The perimeter of a road is obtained by measuring around its section, and in the case of a rectangular road is the combined length of roof, floor, and sides. Thus, a road 7 feet high and $9\frac{1}{2}$ feet wide has a perimeter of $7 + 7 + 9\frac{1}{2} + 9\frac{1}{2} = 33$ feet; and its rubbing surface in square feet, if one mile in length, is $33 \times 1760 \times 3 =$ 174,240 sq. feet.

Area.—The area of a rectangular road is obtained by multiplying the width by the height. Thus a road 6 feet 3 inches high by 7 feet 9 inches wide has an area of 48·4375 sq. feet.

The total pressure on an airway is the pressure per square foot multiplied by the area. If the WG were 1·3 inch, the *total* pressure on the above road would be $1{\cdot}3 \times 5{\cdot}2 \times 48{\cdot}4375 = 327{\cdot}4375$ lbs.

Now, if the airway were only half the area, the pressure per square foot necessary to obtain the same *total* pressure must be doubled; hence it follows that (other conditions being equal) the pressure per square foot necessary to overcome the friction varies inversely as the area.

These first two rules show that the best form for an airway, as regards friction, is the one which has the smallest rubbing

surface in proportion to its area. For example, a road 8 feet square has an area of 64 sq. feet and a perimeter of 32 feet, whilst a road 16 feet by 4 feet has the same area, but has a perimeter of 40 feet. Hence the latter road would offer more resistance to the air than the former in the proportion of 40 to 32. The best form of airway as regards friction is the circular, but usually the shape of the road is governed by practical considerations, such as thickness of seam and character of roof and floor.

Velocity.—The quantity of air passing through a road of given area varies directly as the velocity. If the velocity is doubled, the quantity is also doubled; and therefore double the quantity of air comes in contact with the sides of the airway at double the velocity; this results in four times the friction. If the velocity of the air is increased fourfold, four times as much air comes in contact with the rubbing surface at four times the velocity, so that the increase in friction is $4 \times 4 = 16$. From this we get the rule that the resistance due to friction increases as the square of the velocity. This rule shows the great necessity that exists for keeping down the velocity by the provision of large airways, and by splitting the air.

The Coefficient of Friction.—This is the pressure required to overcome the resistance due to friction for each square foot of rubbing surface exposed to the air when it is travelling at a velocity of 1000 feet per minute. This will be understood by the following example: The pressure required to force air along a road 6 feet by 5 feet and 1200 yards long, at a velocity of 1000 feet per minute, is 15 lbs. per square foot. Find the coefficient of friction.

The total rubbing surface is $(6 + 6 + 5 + 5) \times 1200 \times 3 =$ 79,200 sq. feet. And as the pressure is 15 lbs. per square foot, the total pressure is $15 \times 30 = 450$ lbs. (30 being the area of the road, in square feet). So that a pressure of 450 lbs. is required to overcome the resistance offered by 79,200 sq. feet of surface, when the air moves at a velocity of 1000 feet per minute. Each square foot offers a resistance of $\frac{450}{79200} = 0{\cdot}00568$ lb., which is the coefficient of friction.

In the example given above, the coefficient of friction is expressed in pounds per square foot, but it may also be expressed in inches of water-gauge, or in feet of motive column.

As stated on p. 260, the pressure required to overcome friction varies with the nature of the rubbing surface. The following are coefficients of friction given by various authorities :—

	Pounds pressure per square foot.	Inches of water-gauge.
Atkinson	0·0217	0·00417
Stanley James ...	0·0092	0·00177
Murgue—		
Arched roads ...	0·002392	0·000460
Unlined roads ...	0·005460	0·00105
Timbered roads ...	0·010088	0·00194

The coefficients given by Murgue are averages, and show the influence of the nature of the lining upon the friction.

From these rules the following formula has been arrived at :—

$$pa = ksv^2$$

where p = pressure in pounds per square foot,
a = area of road in square feet,
s = rubbing surface in square feet,
k = coefficient of friction in pounds per square foot,
v = the velocity in thousands of feet per minute.

Both p and k may be expressed in inches of water-gauge or feet of motive column; but care must be taken to employ the same for both.

If the truth of the foregoing rules is accepted, this rule of ventilation is evident; for if p varies as s, and as $\frac{1}{a}$, and as v^2, then it is evident that if we know p, unit rubbing surface, and unit velocity, the pressure in any case is the product of unit pressure or coefficient of friction and $\frac{sv^2}{a}$.

From the formula given above, we have the following:—

$$p = \frac{ksv^2}{a} \quad a = \frac{ksv^2}{p} \quad k = \frac{pa}{sv^2} \quad s = \frac{pa}{kv^2} \quad \text{and } v^2 = \frac{pa}{ks}$$

The following examples show the application of these formulæ:—

(1) Find the water-gauge necessary to force 30,000 cub. feet of air per minute along a road 1800 yards long and 6 feet high by 8 feet wide, taking the coefficient of friction at 0·005 lb. per square foot. By the formula $p = \frac{ksv^2}{a}$,

$$\begin{aligned} k &= 0·005 \\ s &= 28 \times 1800 \times 3 = 151,200 \\ a &= 6 \times 8 = 48 \\ v^2 &= 0·39 \end{aligned}$$

To find the numerical value of v^2, the velocity of the air is first found by dividing the quantity passing by the area, $\frac{30000}{48} = 625$; but this is the velocity in feet per minute, whereas v in the formula represents thousands of feet per minute, therefore $v = \frac{625}{1000} = 0·625$, and $0·625^2 = 0·390625$, which is the numerical value of v^2.

$$\text{Then } p = \frac{0·005 \times 151200 \times 0·390625}{48} = 6·15234$$

This gives the pressure required in pounds per square foot, and must be divided by 5·2 to get the inches of water-gauge—

$$\frac{6·15234}{5·2} = 1·183 \text{ inch W.G.}$$

(2) Find the quantity of air which would circulate the roads in No. 1 example under a pressure of 12·30468 lbs. per square foot.

By the formula, $v^2 = \frac{pa}{ks}$, and, substituting the numerical values in example No. 1—

$$v^2 = \frac{12·30468 \times 48}{0·005 \times 151200} = 0·78122$$

and $v = \sqrt{78122} = 0·8839 = 883·9$ feet per minute

The quantity of air is obtained by multiplying this velocity by the area; $883{\cdot}9 \times 48 = 42427$ cub. feet per minute. Thus by doubling the pressure the quantity is raised from 30,000 to 42,427 cub. feet per minute.

It will be noticed that the increase is not in proportion to the pressure but to the square root of the pressure, and from this we get this important rule:—

The pressure varies as the square of the quantity; and, conversely, the quantity varies as the square root of the pressure. Thus to double the quantity four times the pressure is required, and by doubling the pressure the quantity is increased in the ratio of $\sqrt{1} : \sqrt{2}$, that is as 1 is to 1·4142.

Horse-power to produce Ventilation.—The foot-pounds of work done in ventilation equals weight of air moved multiplied by length of motive column through which it is lifted. That is, foot-pounds = weight of a cubic foot of air × length of motive column × quantity of air in cubic feet.

But the pressure = weight of a cubic foot × length of motive column;

$$\therefore \text{foot-pounds} = \text{pressure} \times \text{quantity}$$

$$\text{and horse-power} = \frac{\text{quantity per minute} \times \text{pressure}}{33000}$$

$$\text{In the first example given, H.P.} = \frac{30000 \times 6{\cdot}152}{33000} = 5{\cdot}5927$$

$$\text{In the second example, H.P.} = \frac{42427 \times 12{\cdot}304}{33000} = 15{\cdot}819$$

These figures show that whilst to produce 30,000 cub. feet of air requires only 5·59 H.P., 15·82 H.P. are required to produce 42,427, the airways being the same. 30,000 bears the same proportion to 42,427 as $\sqrt[3]{5{\cdot}6}$ bears to $\sqrt[3]{15{\cdot}8}$, which proves that the quantity varies as the cube root of the power; thus, by doubling the power the quantity is increased in the proportions of $\sqrt[3]{1}$ to $\sqrt[3]{2}$; that is, as 1 is to 1·259.

Conversely the power varies as the cube of the quantity; thus to double the quantity 2^3 or 8 times the power is required.

The Furnace.—Furnaces are now rarely built to ventilate important collieries, fans being usually preferred. There are, however, many old but large collieries in the North of England and elsewhere which are very efficiently ventilated by furnaces. In many of the old collieries steam is generated underground, and the waste heat from the boilers contributes largely to the ventilation of the mine.

A ventilating furnace in conjunction with underground boilers is probably the cheapest method of ventilating a deep pit, but the advantages are more than balanced by the great inconvenience, and by the element of danger that is introduced by having fires underground.

Fig. 120 shows the ordinary form of ventilating furnace in elevation and plan. The arch over the fire should have an independent firebrick lining which can be replaced when burned out. A cooling drift should be provided on either side through which air can circulate, to prevent the heat from the fire igniting the strata. Large furnaces have fireholes at either side of the firebars as well as in front.

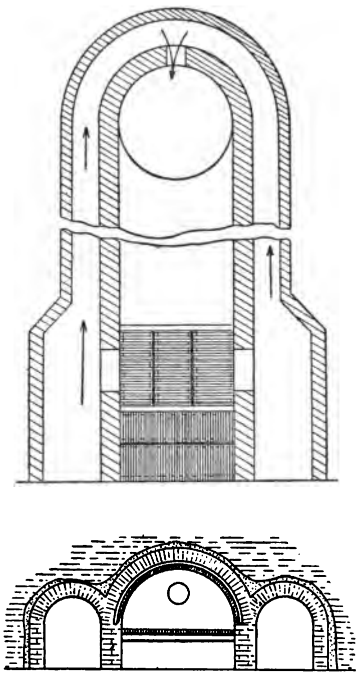


FIG. 120.—Ventilating furnace.

When the mine is gaseous, the return air should not pass over the furnace, but should enter the upcast shaft by a dumb drift, which is a road driven into the shaft some 20 or 30 yards above the furnace, as shown at A, Fig. 121, or the furnace

drift may enter the shaft at a point some distance above the return airway, as shown at B, Fig. 121. When a dumb drift is employed, the furnace must be supplied with the air necessary for combustion by a separate split; and when this is the case, the front of the furnace is closed by brickwork and doors, in order to force the air through the fire, and prevent its being wasted by passing right over the fire.

The pressure- or water-gauge produced by a furnace is calculated as follows:—

A pair of shafts are 1400 feet deep; the temperature in the downcast is 56 degrees Fahr., and in the upcast 161

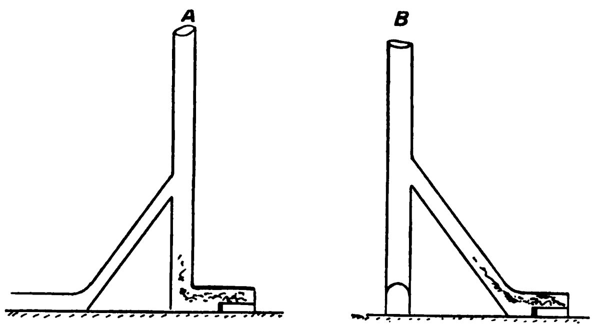


FIG. 121.—Dumb drifts.

degrees. What is the water-gauge? First, find the weight of a cubic foot of air in each shaft by the rule given on p. 257; assume the barometer to stand at 30 inches.

$$\text{In downcast, } W = \frac{30 \times 1{\cdot}3253}{459 + 56} = 0{\cdot}0772 \text{ lb. per cubic foot}$$

$$\text{In upcast, } W = \frac{30 \times 1{\cdot}3253}{459 + 161} = 0{\cdot}0641 \text{ lb. per cubic foot}$$

The difference in weight of a cubic foot of air in each shaft is $0{\cdot}0772 - 0{\cdot}0641 = 0{\cdot}0131$ lb. The depth is 1400 feet, so that the total difference in weight between columns of air 1 sq. foot in area in each shaft is $1400 \times 0{\cdot}0131$ $= \mathbf{18{\cdot}34}$ lbs. This, then, is the ventilating pressure in pounds

per square foot; expressed in inches of water-gauge it is $\frac{18 \cdot 34}{5 \cdot 2} = 3 \cdot 527$ inches W.G.

This calculation shows that the pressure varies directly with the depth of the shafts, and with their difference in temperature; and as the quantity varies with the square root of the pressure, it follows that the quantity varies—

(*a*) As the square root of the difference in the temperature of the two shafts.

(*b*) As the square root of the depth.

For example: If a chimney 60 feet high is built around an upcast pit 250 yards deep, how much per cent. would the quantity of air be increased?

$$\left.\begin{array}{l}\text{As } \sqrt{250} : \sqrt{250 + 20} : : 100 \text{ to} \\ \quad \text{increased quantity}\end{array}\right\} = \frac{16 \cdot 4317 \times 100}{15 \cdot 8114}$$

$$= 103 \cdot 9$$

Ans. 3·9 per cent.

Natural Ventilation.—This occurs when the upcast and downcast shafts are situated at different surface levels, and the temperature of the atmosphere is different from the temperature of the mine.

The temperature of the strata is nearly constant at a depth of about 60 feet, and averages about 50 degrees; it increases at the rate of about 1 degree for every 60 feet of descent. In summer the natural temperature of a shallow mine is less than that of the atmosphere, and in winter the atmosphere is the cooler.

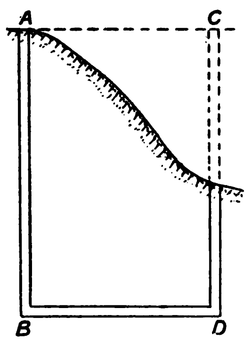


FIG. 122.—Natural ventilation.

In Fig. 122 the two columns of air are AB and CD; part of CD is in the atmosphere, so that in winter the column CD will be cooler and heavier than AB. This will give rise to ventilating pressure, and CD will be the downcast. In summer the

conditions will be reversed, and the air in AB will be the cooler and heavier, which will result in AB being the downcast. When the temperatures of atmosphere and mine are equal, the two columns of air will balance, and no ventilation will take place. Natural ventilation is often employed in metal mines, but is never relied on for collieries, except when they are on the smallest scale. The ventilation of a deep mine is, however, always assisted by natural ventilation, because the mine temperature is higher than the temperature of the atmosphere; as the outside temperature rises the natural ventilation slackens, so that in summer increased ventilation is required.

Steam Jets.—For temporary purposes steam jets may be employed to heat the air in the upcast shaft, and so produce ventilation. A steam-pipe is taken part way down the shaft; horizontal pipes branch out at the bottom, and the steam issues from perforations bored in their upper surface.

Ventilating Fans.—Fans may either force fresh air down the downcast shaft, or draw the return air from the upcast; the latter is by far the more common, as it is inconvenient to close the top of the downcast shaft, and therefore exhaust fans are almost universal. The fan is connected to the upcast shaft by a drift, and the top of the shaft is kept closed, so that all the air coming to the fan has to pass through the mine.

Ventilating fans depend for their action upon centrifugal force. If a stone is whirled round in a circle at the end of a string, tension is put upon the string, and if it is cut, the stone flies off at a tangent. Similarly, if air is whirled round in a circular box in which vanes revolve, it tends to fly off at the circumference, and is only prevented from doing so by the box. This tendency to fly off at a tangent, which is known as centrifugal force, puts pressure on to the air near the circumference of the box, and reduces the pressure of the air at its centre; so that if an opening is cut in the circumference of the box, the air will rush out, owing to the increased

pressure at that point, and if another aperture is cut in the centre, air rushes in, owing to the reduced pressure at that point; this results in a constant flow of air through the fan. What a fan really does is to drive the air from its centre to its circumference; the air in the fan body and in the roads connected to it is thereby rarefied and made lighter than the air in the downcast shaft, thus giving rise to a motive column.

The theoretical motive column, or water-gauge, produced by a fan depends upon the velocity of the blade tips, and is calculated by the following formula:—

$$h = \frac{v^2}{g}$$

where h = air column in feet,
v = velocity of blade tips in feet per second,
g = force of gravity = 32·2.

Example.—What water-gauge should be produced by a fan 30 feet in diameter when making 60 revolutions per minute?

$$\text{Circumference of fan} = 94{\cdot}24 \text{ feet}$$

$$v = \frac{94{\cdot}24 \times 60}{60} = 94{\cdot}24$$

$$\text{and } v^2 = 8881{\cdot}2$$

$$\text{Then } h = \frac{8881{\cdot}2}{32{\cdot}2} = 275{\cdot}8 \text{ feet}$$

The weight of a cubic foot of air in the upcast must next be calculated by the rule already given. Suppose this works out at 0·07 lb., then the total weight of the air column, 1 sq. foot in area, is $275{\cdot}8 \times 0{\cdot}07 = 19{\cdot}3$ lbs., and $\frac{19{\cdot}3}{5{\cdot}2}$ = 3·7 inches of water-gauge.

No fans actually produce their theoretical water-gauge; the ratio of actual water-gauge produced to the theoretical water-gauge is known as *manometric efficiency*. If, for example, the above fan actually produced a water-gauge of 2·6 inches, its manometric efficiency would be $\frac{100 \times 2{\cdot}6}{3{\cdot}7} = 70{\cdot}27$ per cent.

Types of Fans.—Fans may be either open running or enclosed. In the former the vanes and casing revolve together, and the casing is open at the circumference; in the latter the casing is stationary, and provided with an opening for the escape of the air. Fans may also have single or double inlets. Single-inlet fans take the air in on one side only, whilst double-inlet fans communicate with drifts at either side. The drifts for a double-inlet fan are shown in Fig. 123. Fans of this type are usually divided by a plate or diaphragm as

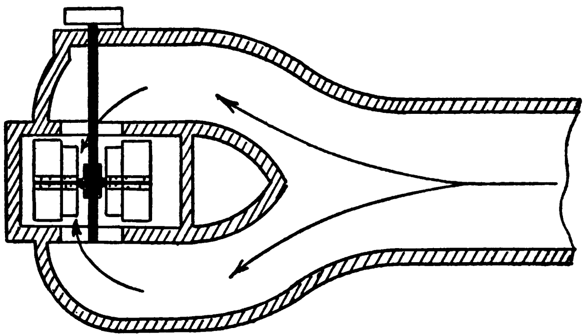


FIG. 123.—Double-inlet fan.

shown, so that the air from the two drifts does not meet in the fan and set up cross-currents.

Fans may be either of large diameter and run slowly, or of small diameter and run quickly; the tendency is now towards the latter type. Quick-running fans may be either driven direct by high-speed engines, or by belts or ropes by low-speed engines; the latter being the more common. Usually duplicate engines are provided in case one engine requires repairs or breaks down.

The Schiele Fan.—These fans have double inlets, and are made of small diameter to run at a high speed. As shown in Fig. 124, they have wings, curved backwards and riveted to a central disc or diaphragm. The fan is placed eccentrically in a spiral casing, so that the velocity of the air is

gradually reduced from the time it leaves the blades to when it is discharged into the atmosphere. Almost all fans have an expanding chimney, for if the air from the fan is discharged at a high velocity into the atmosphere, an unnecessary amount of power has to be expended.

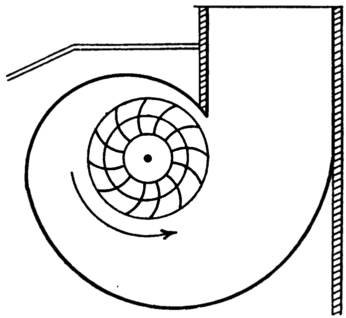


FIG. 124.—Schiele fan.

The blades taper in width towards the circumference, so that the air passes through the fan at a uniform velocity; if the air leaves the fan blades too slowly, it may re-enter the fan instead of being swept clear away. These fans are generally driven by belts or cotton ropes from the fan engine.

Schiele fans are made up to about 16 feet in diameter; a 14-feet fan will produce about 250,000 cub. feet of air at a 6- or 7-inch water-gauge.

The Guibal Fan.—This is one of the oldest types of centrifugal ventilators. It is a single-inlet fan, and is made up to about 50 feet in diameter by about 12 feet in width. The blades, which are eight or ten in number, are of timber, and bolted to an angle-iron framework; this is bolted to a set of two or three cast-iron centres which are keyed on to the fan shaft. The blades are straight, but are set back, not fixed radially. The casing and evasée chimney are of brick-work; the casing fits closely to the fan for about three parts of its circumference, the air being discharged into the evasée chimney through an adjustable wooden shutter. There are many of these fans now at work, but no new ones are built upon this principle, as they require very costly foundations and buildings, and are of somewhat weak construction on account of the many bolts which are apt to work loose.

Walker's Guibal.—This is a modified Guibal fan; it is built entirely of iron and steel, has a double inlet and blades

curved backwards. It is made smaller than the original Guibal, and runs much more quickly. The anti-vibration shutter is a feature of this fan. The opening in the shutter is in the form of an inverted V; by this arrangement each blade is shut off gradually from the evasée chimney. With the ordinary shutter the delivery from each blade ceases the moment the blade passes the shutter; this causes sudden variations in pressure, and sets up great vibration, which may end in a breakdown.

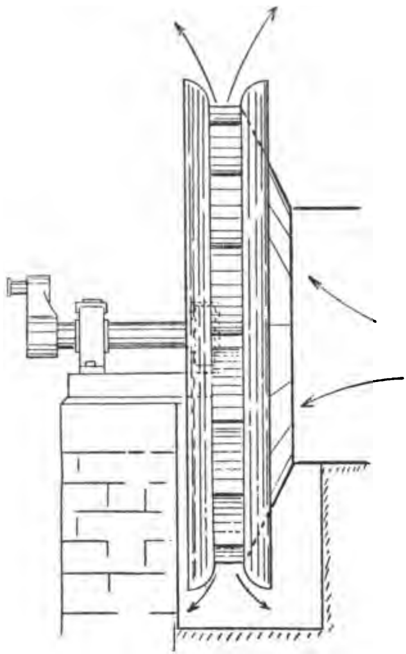


FIG. 125.—Waddle fan.

The Waddle Fan.—This is a single-inlet, direct-driven open-running fan; it requires little brickwork, as it has no evasée chimney.

The fan casing is riveted to the blades and revolves with them, the air being discharged all round the circumference. Formerly these fans were made of large diameter (up to 45 feet) and revolved comparatively slowly, but now they are being made smaller in diameter with a corresponding increase in speed. For quantities of air up to 350,000 cub. feet per minute the new fans are made self-contained, and have no bearings in the fan drift. The casing is carried beyond the blades and belled out, in order to enable the fan to deliver the air into the atmosphere at a reduced velocity; this serves the same purpose as an evasée chimney. Fig. 125 shows the self-contained Waddle fan in end elevation.

It will be noticed that its width decreases from the centre to the circumference, so that the air passes through the fan at a uniform velocity; the blades are curved back into the circumference. The fan is hung at the end of the shaft, and is carried by bearings on the engine-bed only; this arrangement leaves the inlet perfectly open and unrestricted.

A 21-feet Waddle fan gave the following results:—

136 revs. per min.	88,700 cub. ft. per min.	5·2 in. W.G.
168 „ „	110,200 „ „	7·4 „

The Capell Fan.—These fans are made of very varying widths and diameters; they have both single and double inlets, and are sometimes driven direct, but more often by belts.

As shown in Fig. 126, they consist of an inner cylinder fitted with curved blades, which discharge the air into an

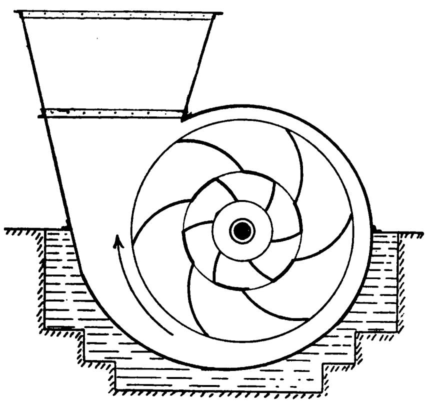


FIG. 126.—Capell fan.

outer cylinder, also fitted with curved blades, from whence it is driven into the expanding chimney.

The air which passes through the fan is acted on, first by the inner and then by the outer blades. The newer fans are provided with scoops in their inlets in addition to the two sets of blades. These fans are at work at many places, and can

be built to give very large quantities of air at extremely high water-gauges.

Size of Engines to drive Fans.—The size of a simple engine to drive a fan is calculated as follows:—

Find the size of an engine to drive a fan giving 150,000 cub. feet of air per minute at a water-gauge of $3\frac{1}{4}$ inches; the efficiency of the plant being 65 per cent., and the average steam pressure on the piston 45 lbs. per square inch. The foot-pounds of work done on the air per minute are $150{,}000 \times 3{\cdot}25 \times 2 = 2{,}535{,}000$.

As the efficiency of the plant is 65 per cent., the foot-pounds developed by the engine must be—

$$\frac{2535000 \times 100}{65} = 3{,}900{,}000 \text{ foot-pounds per minute}$$

The foot-pounds developed by the engine are—piston speed in feet per minute × pressure per square inch on piston × area of piston. So that, taking the piston speed to average 400 feet per minute, the area of the piston is—

$$\frac{3900000}{400 \times 45} = 216{\cdot}6 \text{ sq. inches}$$

and the diameter is $\sqrt{\frac{216{\cdot}6}{0{\cdot}7854}} = 16{\cdot}6$ inches

Or the calculation can be done in one operation; thus—

$$\sqrt{\frac{150000 \times 3{\cdot}25 \times 5{\cdot}2 \times 100}{65 \times 400 \times 45 \times 0{\cdot}7854}} = 16{\cdot}6 \text{ inches}$$

Closing Upcast Pit Tops.—When an upcast shaft is used for winding it must be closed by some arrangement of doors, in order to prevent the fresh air from being drawn into the fan drift. One method of doing this is shown in Fig. 127. The pit top is enclosed by a casing of iron, brick, or timber, the entrance into the enclosed space being closed by doors *d, d*. Room is provided between the doors on each side of the pit for as many corves as the cages carry. Whilst the cages are in the shaft, the outer doors are opened for the passage of the empty and full corves to and from the air-locks; and when

the corves on the cages are to be changed, the inner doors are opened and the outer doors closed.

Ventilating the Workings.—The air is conducted round the workings by means of stoppings, doors, overcasts,

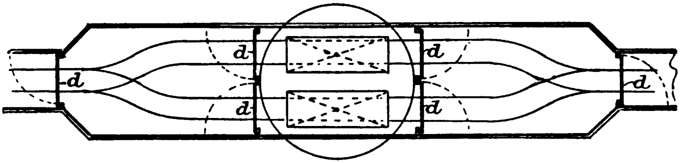


FIG. 127.—Upcast pit top, closed with double doors.

brattice, and air-pipes. Fig. 128 shows the manner in which these are employed.

The roads closed by the stoppings are permanently closed; doors are employed in roads which have to be closed to the

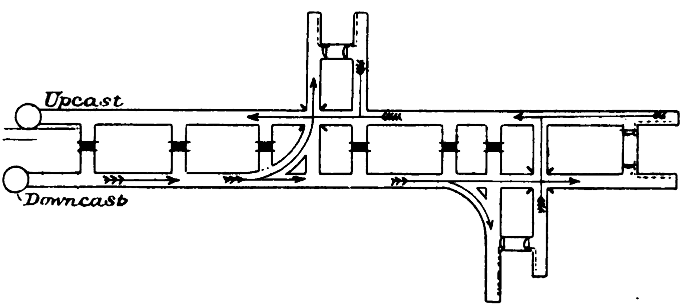


FIG. 128.—Distributing the air-current.

air, but open for traffic; overcasts are used when one current of air has to cross another without mixing with it.

The dead-ends are ventilated by brattice or air-pipes.

Stoppings.—These usually consist of brick walls 9 inches in thickness; sometimes two walls are built a few feet apart, and the space between is filled in tightly with dirt.

Doors.—Ventilating doors should be set in pairs, the space

between them being long enough to admit the corves and pony, so that when one door is opened the other is closed. They should close automatically, and should be set to open against the current. When it is required to reduce the ventilation passing through a district, regulating doors are employed; they are similar to the ordinary doors, but are provided with an opening, the area of which can be regulated by an adjustable slide. In districts which are newly opened out, regulators are obliged to be used, but the best method of regulating the quantity of air in each district is to adjust the length of each split.

Air-crossings.—A section through an air-crossing or overcast is shown in Fig. 129.

The roof is blown down to make sufficient height, and the two brick side walls *a*, *a* are built on either side of the lower

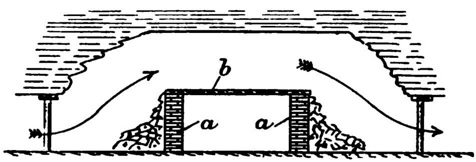


FIG. 129.—Air-crossing.

road. Planks, *b*, are then laid across the road from wall to wall. The joints between the planks are made air-tight by nailing thin strips of wood over them. This type of air-crossing is very common, but it is neither very strong nor completely air-tight, and would be blown out if an explosion occurred. A stronger, though much more expensive air-crossing is made by arching one road and taking the air across it through a short stone drift driven a few yards above or below.

Bratticing.—Dead-ends, such as the ends of headings beyond the last slit and banks in pillar-and-stall workings, are ventilated by brattice or by air-pipes. A brattice is a partition dividing the road longitudinally into an intake and return. For unimportant roads, brattice cloth is commonly

employed, and, for the more important roads, such as stone drifts, which may have to be driven great distances without communications, a brick wall is built from roof to floor. The brattice is not put down the centre of the road, but about a couple of feet from one side, the narrow side being used as an intake and the wider side as a return, and for the traffic.

Air-pipes of wood or iron are frequently used instead of brattice; they are cheaper, and take up less room, and can be easily fixed or taken down.

Splitting the Air.—At one time the whole of the ventilation was taken round the whole of the workings in a single current; this had many serious disadvantages, indeed it would be quite impossible to ventilate a modern colliery in this manner. The air is now always split, each split serving a separate district. There are usually several main roads leading from the downcast shaft, and part of the air is led along each. These form the main splits; each main split is then divided into sub-splits, each of which ventilate one district. Not more than from eighty to a hundred men should work in one split.

The advantages of splitting the air are—

(1) Each district is ventilated by fresh air.

(2) A fall of roof only affects the district in which it occurs.

(3) The smoke from a fire, or after-damp from a small explosion, would not be carried through the whole of the workings.

(4) The total quantity of air is increased.

(5) The current velocity is kept within reasonable limits.

If splitting is carried too far, the velocity of the air in each district may be reduced until the current is not brisk enough to sweep away any gas which may be made. The current velocity should not be too great, as that would add to the danger of a coal-dust explosion, and increase the risk of a defective lamp firing an explosive mixture; moreover, an extremely high velocity is unpleasant, and adds greatly to the friction.

A fair velocity for the air to travel at is—

In shafts, up to about 1000 feet per minute.

In main airways, up to from 800 to 1000 feet per minute.

In working places, from about 150 to 300 feet per minute.

The quantity of air per man per minute varies very greatly in different mines, and may be anything from about 100 to about 500 cub. feet.

CHAPTER XXI.

INSTRUMENTS.

The Barometer.—This is an instrument for measuring the pressure of the atmosphere. In the ordinary mercurial barometer the pressure of the atmosphere is balanced against the pressure due to a column of mercury. In Fig. 130 an ordinary syphon barometer is shown. The tube *a* is about 36 inches long; it is closed at the top, and the bottom end is turned up and terminates in a small vessel, as shown. If this tube is filled with mercury and raised to a perpendicular position, the mercury will fall in the tube until its weight is exactly balanced by the atmosphere. If the atmosphere had no weight, the mercury would fall in the tube and run out over the top of the vessel; but it is prevented from doing this by the atmosphere pressing upon the mercury in the vessel. The atmospheric pressure per square inch must be exactly the same at *b* (Fig. 130) as the mercurial pressure at *c* (the mercury below *bc* balances itself), so that if we know the pressure at *c* we also know the pressure of the atmosphere.

When the atmospheric pressure decreases the mercury falls in the tube, because the lessened atmospheric pressure is unable to balance so great a pressure of mercury.

One cubic inch of mercury weighs 0·4908 lb. Hence, height of barometer in inches multiplied by 0·4908 gives atmospheric pressure in pounds per square inch.

Example.—Find atmospheric pressure when the barometer stands at 29·15 inches.

$$29{\cdot}15 \times 0{\cdot}4908 = 14{\cdot}308 \text{ lbs. per square inch}$$

The barometer is also used to indicate differences in level. As we descend a pit the column of air above us becomes longer, and consequently heavier, and if a mountain is ascended, part of the atmosphere is left below, so that the column above is shorter and lighter. It follows, therefore, that the barometer rises with descent, and falls with ascent.

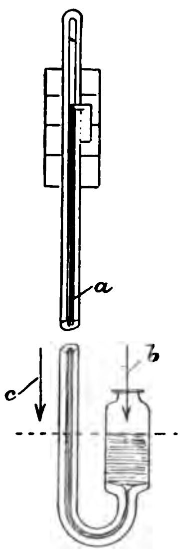


FIG. 130. Barometer.

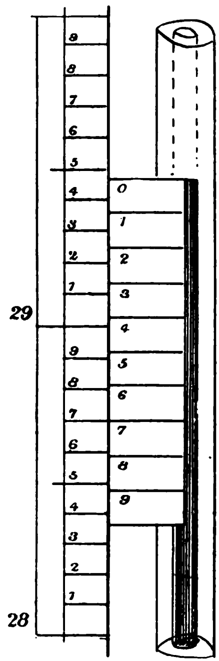


FIG. 131.—Barometer scale and vernier.

This variation amounts to about 1 inch for every 900 feet of rise or fall; so that a barometer at the bottom of a shaft 600 yards deep would read about 2 inches higher than a similar instrument placed on the pit top.

The height of the barometer varies at sea-level between about $28\frac{1}{2}$ and $30\frac{1}{2}$ inches.

The height at which the mercury stands is read off by means of a fixed scale and sliding vernier. The scale is divided into inches and tenths, and the tenths are subdivided into hundredths by the vernier. Fig. 131 represents a portion of the face of a barometer, and shows the fixed scale and vernier in position. The vernier is $1\frac{1}{10}$ inch long, and is divided into ten equal parts, and the fixed scale is divided into inches and tenths, so that each division on the vernier equals $1\frac{1}{10}$ division on the fixed scale. The fixed scale is marked upwards and the vernier downwards. To read the height of the mercury the top of the vernier is set exactly level with the top of the mercury. Supposing the mercury reaches exactly $\frac{4}{10}$ of the distance between two of the divisions on the scale, the line marked 4 on the vernier will exactly coincide with one of the division lines on the scale. The line marked 0 on the vernier, being level with the mercury, is exactly $\frac{4}{10}$ above one of the division lines on the scale; line 1 on vernier is $\frac{3}{10}$ above the next line on scale, because the divisions on the vernier are longer by $\frac{1}{10}$ than the divisions on the scale; line 2 on vernier $\frac{2}{10}$ above; line 3, $\frac{1}{10}$ above; and line 4 exactly level with a division line on scale. From this we get the rule to read the vernier and scale, which is—

First read the inches and tenths from the fixed scale (reading upwards), then notice which division line on the vernier exactly coincides with a mark on the fixed scale; read that off from the vernier as hundredths.

In the figure the inches and tenths are 29·4, and the 7 on the vernier exactly coincides with a line on the scale so that the hundredths are 7, hence the reading is 29·47 inches.

It is important to understand the principle upon which verniers are constructed, as they are largely employed in mathematical and surveying instruments.

Aneroid Barometers.—No mercury is employed in aneroid barometers. They can be made very sensitive, and are much more portable than mercurial barometers, but more liable to disarrangements. Fig. 132 shows a simple arrangement of a self-recording aneroid. *a* is a corrugated steel box partly

exhausted of air. The atmospheric pressure upon the upper surface of the box tends to push it down, whilst the spring of the box tends to push it up. When the atmospheric pressure increases the box top is depressed, and when it decreases it is raised by the spring.

One end of the pointer *b* is connected to the box top by the link *c; b* is pivoted at *d*, and carries a pencil, *e*, at the end. The pencil-point presses lightly against a paper wrapped round the slowly revolving drum *f*.

When the atmospheric pressure increases the box top is depressed and the pencil is raised; the combined vertical movement of the pencil and horizontal movement of the drum

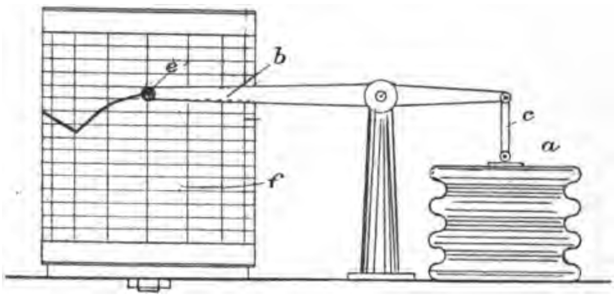


FIG. 132.—Aneroid barometer.

result in a slanting line being drawn upon the paper. In this way a chart is made which shows continuously the variations in atmospheric pressure, the quicker the variations in pressure the more the lines on the chart approach the vertical.

Thermometers.—Moderate temperatures are measured by thermometers, but when the temperatures are very high, pyrometers are employed. Thermometers depend for their action upon the fact that certain substances vary greatly and uniformly in bulk with the temperature. The ordinary thermometer consists of a thick glass tube of small but uniform bore, sealed at one end and terminating in a bulb at the other.

This bulb contains alcohol or mercury, which also extends a short distance up the tube.

As the temperature increases the mercury expands in the bulb and rises in the tube, and when the temperature decreases the converse is the case.

The two fixed points of temperature are the freezing and boiling points. Freezing-point is the temperature of melting ice, and the boiling-point is the temperature of steam from water boiling under the normal atmospheric pressure. (The temperature at which water boils varies with the pressure, being less when the pressure is low.)

Thermometers are marked on three different scales, namely, Centigrade, Fahrenheit, and Reaumur. The gradations on each are—

Fahrenheit,	freezing-point		$32°$,	boiling-point		$212°$
Centigrade,	,,	,,	$0°$,	,,	,,	$100°$
Reaumur,	,,	,,	$0°$,	,,	,,	$80°$

From this it follows that—

$1°$ Fahr. $= \frac{5}{9}°$ Cent. and $\frac{4}{9}°$ Reaumur
$1°$ Cent. $= \frac{9}{5}°$ Fahr. and $\frac{5}{4}°$ Reaumur
$1°$ Reaumur $= \frac{9}{4}°$ Fahr. and $\frac{4}{5}°$ Cent.

Fahrenheit's scale is popularly employed in Great Britain, but for scientific work the Centigrade scale is almost universal.

To convert degrees Cent. to degrees Fahr. multiply by $\frac{9}{5}$ and add 32.

Example.—Convert 60 degrees Cent. to degrees Fahr.

$$60° \text{ Cent.} = \frac{60 \times 9}{5} = 108° \text{ Fahr.}$$

So that the temperature is 108 degrees above freezing-point on Fahrenheit's scale, and as the freezing-point is 32 degrees, the reading will be $108 + 32 =$ **140 degrees.**

Convert 162 degrees Fahr. to degrees Cent.

162 degrees Fahr. = 162 degrees − 32 degrees = 130 degrees above freezing-point, and $130 \times \frac{5}{9} =$ **72·2 degrees Cent.**

Examples similar to the above may be worked by the following formulæ :—

$$\text{Degrees C} = \frac{(F - 32)5}{9}$$

$$,, \quad F = \frac{C \times 9}{5} \text{ and } + 32 \quad \frac{R \times 9}{4} + 32$$

$$,, \quad R = \frac{(F - 32)4}{9}$$

The Water-gauge.—Small differences in fluid pressure are measured by a "water-gauge." This, in its simplest form, consists of a glass tube bent into the shape of the letter U, and provided with a sliding scale of inches and decimals. Water is poured into the tube, filling the bend and reaching a little way up each leg. The top of one leg is open to the atmosphere and the other communicates by a pipe with the air, the relative pressure of which it is desired to measure.

The water-gauge is used in mines to measure the pressure which produces ventilation. As explained in Chapter XX., mines are ventilated by a difference in pressure between columns of air in the upcast and downcast shafts, the pressure in the downcast shaft being always the greater. If a water-gauge is placed in the intake and the pipe from the closed end is led into the return, one column of water is subjected to the pressure of the intake and the other of the return air. But as the pressure of the intake is the greater, it will press down the water in the leg with which it communicates, raising it in the other leg, and the difference in pressure per square foot is equal to a column of water 1 square foot in area, and of a height equal to the difference in level between the water in the two legs of the water-gauge.

A cubic inch of water weighs 0·036 lb., so that a column of water 1 square foot in area and 1 inch high weighs $0{\cdot}036 \times 144 = 5{\cdot}2$ lbs.

This gives the rule that: Inches of water-gauge $\times$ 5·2 = pounds pressure per square foot.

For example: If a water-gauge is placed in a fan engine-house, and one leg communicates by a pipe with the fan drift, the water in the open end will be depressed; the difference in level is found by sliding the scale by means of the screw until zero on the scale is level with the lower column of water, and then reading the scale off at the level of the water in the other leg. Suppose the reading to be 3·15, the fan is then producing a reduction in pressure of $3{\cdot}15 \times 5{\cdot}2 = 16{\cdot}38$ lbs. per square foot.

Fig. 133 shows an improved form of water-gauge. It consists of a box divided vertically by a partition, communication between the divisions being made by the small brass tube *a*. One division is open to the intake, and the other communicates with the return by a pipe in the usual way. The difference in height of the two columns of water is read off by the scale *b*, which is adjustable by means of a screw. The advantage of this form of water-gauge is that it can be read more accurately than the ordinary form, because the water does not oscillate to the same extent. This is due to the comparatively large area of water in each division, and to the restricted area of the brass tube *a*. Other advantages are that the level of the water in the rectangular chambers can be more easily determined than in the case of the smaller glass tubes, and, if necessary, the tap in the tube *a* can be closed, and the water-levels preserved for subsequent verification.

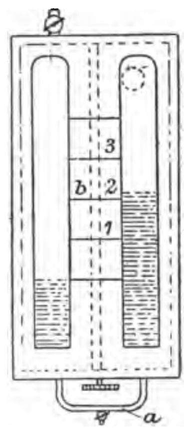


FIG. 133.—Water-gauge.

When the end of the pipe leading from the water-gauge is in movable air, as is usually the case, an incorrect pressure is obtained unless the pipe end is well shielded from the current. This is best done by bending the pipe at right angles to the air-current, and packing the end with felt or cotton wool.

The Hygrometer.—The humidity of the air is measured by an instrument known as the hygrometer.

Air can only hold a certain amount of moisture in suspension, and this amount varies with the temperature, the higher the temperature of the air, the more moisture it will carry. When air contains its maximum amount of moisture it is said to be saturated. If saturated air is cooled, its capacity for moisture is decreased, and part of the moisture it contains is deposited in the form of dew. On the other hand, if the temperature of air is increased, its capacity for moisture is also increased, and it dries up any moisture it comes in contact with. Thus cool air goes down a pit, and becomes heated as it passes along the roads; this increases its capacity for moisture, and causes it to absorb any there may be present. This drying action has a very important bearing upon the danger of explosions from coal-dust.

The most common form of hygrometer is that known as Mason's, which depends upon the fact that the action of evaporation is accompanied by a lowering of the temperature.

This hygrometer consists of two thermometers placed side by side. The bulb of one of these is kept moist by being wrapped round with a piece of muslin. One end of this muslin is dipped in a small vessel of water, and draws the water up to the bulb by capillary attraction. If the atmosphere is dry, moisture is evaporated from the muslin round the bulb, causing the temperature to be lowered, so that the reading of the two thermometers is different. If the air is saturated with moisture, no evaporation can take place, so that the reading of the two thermometers is the same.

Anemometers.—The velocity at which air travels is measured by an anemometer. A Davis anemometer is shown in Fig. 134.

It consists of a small fan, the vanes of which, being set at an angle, are revolved by the air. These vanes are connected by gearing to dials, which show the number of lineal feet of air which pass the instrument.

To read an anemometer similar to the one shown in Fig. 134, the position of the small finger is first noted, this gives the hundreds; the tens and units are next read off from the larger dial. The reading of the instrument shown in the figure is 678.

FIG. 134.—Davis anemometer.

To use the anemometer, it is held in an airway for exactly one minute, the dials are read off before and after the trial, and the former reading subtracted from the latter gives the number of lineal feet of air which have passed the instrument during the minute, or, in other words, the velocity of the current in feet per minute. The velocity in feet per minute multiplied by the area of the road gives the quantity passing in cubic feet per minute.

Example.—A road is $8\frac{1}{2}$ feet wide by 7 feet high; the velocity is 550 feet per minute. What is the quantity?

$$\text{Area of road, } 8{\cdot}5 \times 7 = 59{\cdot}5 \text{ sq. feet}$$
$$\text{Quantity passing, } 59{\cdot}5 \times 550 = 32{\cdot}725 \text{ cub. feet per minute}$$

If the quantity of air passing along a road is known, the velocity can be determined by dividing the quantity by the area of the road.

Example.—The quantity of air passing along a road of the form and dimensions given in Fig. 135 is 35,000 cub. feet per minute. What is the velocity?

The area of the road must first be determined. It will be noticed that a section of the road consists of a semicircle and a rectangle; the areas of each must be found separately, and added together.

The area of the semicircular portion is equal to half the area of a circle $7\frac{1}{2}$ feet in diameter. The rectangular portion is $7\frac{1}{2}$ feet wide, and its height is the total height of the road minus the radius of a circle $7\frac{1}{2}$ feet in diameter, that is, $8 - 3\frac{3}{4} = 4\frac{1}{4}$ feet.

sq. feet.

$$\text{Area of semicircular portion} = \frac{7{\cdot}5^2 \times 0{\cdot}7854}{2} = 22{\cdot}089$$

$$\text{,, rectangular portion} = 7\tfrac{1}{2} \times 4\tfrac{1}{4} = \underline{31{\cdot}875}$$

Total area ... ... 53·964

$$\text{Velocity} = \frac{\text{quantity}}{\text{area}} = \frac{35{\cdot}000}{53{\cdot}964} = 648{\cdot}6 \text{ feet}$$

The velocity of an air-current varies greatly in different parts of the same section of a road; it is always higher in the centre of the road, and lower near the roof, floor, and sides.

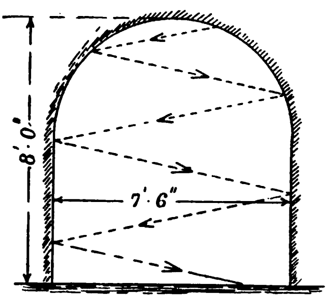


FIG. 135.--Positions of anemometer in measuring the current.

In some cases it has been found that the velocity in the centre is more than double what it is at the sides. This is due to the air which rubs against the sides being retarded by the friction against them. If the anemometer is held in one position during the whole of the trial, erroneous results are obtained. To obtain the average velocity, it should be slowly moved in the manner indicated by the dotted lines and arrows in Fig. 135. In making very important observations, the road should be divided by vertical and horizontal strings, and the velocity obtained in each of the divisions thus made.

Care must be taken to keep the anemometer squarely facing the current; and the section of the road in which it is held should be as regular as possible, because ledges and projections produce eddies, and increase the variations in velocity.

CHAPTER XXII.

LIGHTING.

WHERE naked lights are permitted they generally take the form of tallow candles or small tin oil lamps; the former are the more generally used, being stuck into lumps of clay, by which they can be fixed to the props or to the coal face. Oil lamps are chiefly used in Scotland; they consist of small conical-shaped tin vessels, having a spout through which the wick passes and a hook on the top, by which they can be secured to the timbers, or carried on the workman's cap. The employment of naked lights is becoming rarer as collieries are increasing in size and depth; in some districts naked lights are commonly used, whereas in others safety-lamps are almost universal.

Naked lights give a better light than the best safety-lamps, hence where they can be used the danger from falls of roof is somewhat reduced, and the output of coal per man is slightly increased.

The downcast pit bottom and sidings are illuminated either with large paraffin lamps, gas, or electric light; the latter is now the most common at large collieries, as it is safe in an explosive mixture, except in the case of accident. A further advantage of electric light at pit bottoms is that open lights of any sort can be altogether prohibited down the pit, and this lessens the chance of their being inadvertently taken into the workings.

Safety-lamps.—A safety-lamp should be absolutely incapable of firing an explosive mixture under any possible combination of circumstances which could occur underground;

it should give a fair light throughout the whole shift, even when used in districts where the air-current is sluggish and impure, and it should be simple in design and easy to clean.

The principle upon which all safety-lamps depend is due to Sir Humphrey Davy, who discovered that pit gas could not be fired by a flame which had passed through metal tubes of certain length and diameter, because the cooling action of the metal lowered the temperature of the flame below that required to ignite fire-damp. Subsequent experiments showed that the same result was achieved by passing flame through a gauze constructed of wires about $\frac{1}{50}$ inch in diameter and having 784 apertures per square inch.

Safety-lamps, then, depend for their security upon the fact that fire-damp will not explode until exposed to a certain temperature, which is known as the temperature of ignition, and that when a flame has passed through a wire gauze it is cooled down by the wires to below this temperature of ignition.

A light enclosed in a wire gauze may ignite gas under the following conditions :—

1. By reason of an internal explosion. If the internal volume of the lamp is large, and it is allowed to become filled with an explosive mixture, an internal explosion might occur and be of sufficient violence to force the flame through the gauze at a high velocity, in which case the flame is not exposed to the cooling action of the gauze for a sufficient length of time to reduce the temperature to the necessary point. An explosion may be caused when the flame passes through the gauze at a velocity of 6 feet per second. This danger is overcome in modern lamps by restricting the outlet for the products of combustion so as to keep the upper part of the lamp filled with incombustible gases; this prevents a dangerous accumulation of an explosive mixture in the lamp, and if an internal explosion should occur, it is quickly extinguished by lack of oxygen.

2. By the flame being blown through the lamp when it is exposed to a current travelling at a high velocity. The older

lamps are unsafe when exposed to high velocities; the modern lamps are provided with shields or bonnets, and have their inlets and outlets so arranged as to prevent a rapid current of air passing *through* the lamp even when it is exposed to a very high velocity.

3. By the gauze being allowed to become red hot, and so lose its cooling action. This may occur where lamps are left in an explosive mixture, which continues to burn in them. A good safety-lamp should become extinguished if left in an explosive mixture, the products of combustion should not be able to pass freely away; hence the lamp becomes filled with incombustible gases, and the flame dies out.

The Davy Lamp.—This lamp is shown in Fig. 136. It consists of a brass oil-vessel, surrounded by a wire gauze about 7 inches long and $1\frac{1}{4}$ inch in diameter. The vessel is screwed on to a brass ring, which is connected to the lamp top by four bars or standards. The course of the air to and from the lamp is indicated by arrows in the figure. The Davy lamp is now quite obsolete. It gives a poor light, owing to the flame being surrounded by gauze, and is unsafe in a current which has a velocity of more than from 3 to 5 feet per second.

When placed in a tin case with a glass front, it is known as a "Tin can Davy," and in this form is still employed by the officials in some of the North of England pits on account of its alleged virtues as a gas-tester.

The Clanny.—In this lamp (Fig. 137) the flame is surrounded by a short thick glass cylinder surmounted by a gauze. The fresh air enters above the glass, and the products of combustion pass out at the top. Owing to the absence of gauze around the flame, the Clanny gives a superior light to the Davy; but it is not now used, being unsafe at a velocity of about 8 or 10 feet per second.

The Stephenson.—This lamp (Fig. 138) has a gauze rather larger in diameter than that of the Davy; the gauze surrounds a glass cylinder provided with a perforated copper cap. The air to support the flame enters through holes in the base of the lamp, and the products of combustion pass out through

the top of the glass. The inlet holes are about $\frac{1}{50}$ inch in diameter, and are rather apt to become clogged with dust.

The Stephenson lamp will withstand a current of from 8 to 10 feet per second, its superiority over the Davy in this respect being due to the glass cylinder, which acts as a shield and prevents the flame being blown straight through the gauze. The long glass cylinder is open to the objection that if an internal explosion should occur the cylinder acts like a cannon,

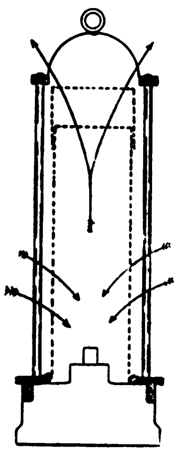


FIG. 136.—Davy lamp.

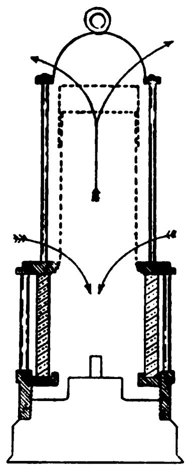


FIG. 137.—Clanny lamp.

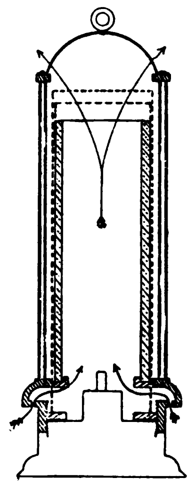


FIG. 138.—Stephenson lamp.

and projects the flame through the top of the glass at a high velocity.

The three lamps just described are now never used, as they do not comply with that clause in the Coal Mines Regulation Act which provides that all safety-lamps must be so constructed as to be safely carried against the air-current ordinarily prevailing in that part of the mine in which they are used, even if such current should be inflammable.

The Marsaut.—Many of the lamps now in use belong to some modification of this type of lamp. Fig. 139 shows the

Marsaut in its ordinary form. It is similar in construction to the Clanny, except that it is provided with a shield or bonnet, and is fitted with two, and in some cases three gauzes. The gauzes are slightly conical in shape, as shown in the figure. The course of the air-current to and from the lamp is indicated by the arrows; there is no distinct division between the intake and return air, which is detrimental, as the fresh air is apt to mingle with the spent air, causing the light to burn dimly.

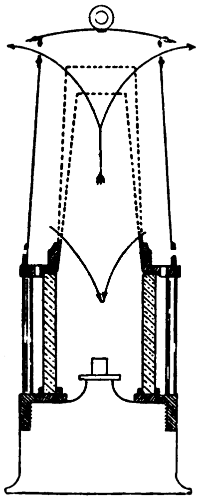


FIG. 139.—Marsaut lamp.

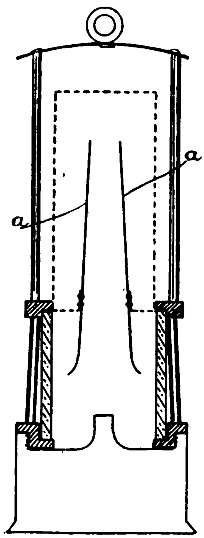


FIG. 140.—Mueseler lamp.

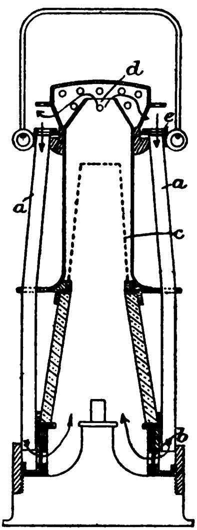


FIG. 141.—Hepplewhite-Gray lamp.

A lamp such as the one shown in Fig. 139, can withstand an explosive mixture travelling at a velocity of from 30 to 40 feet per second.

The Mueseler.—The chief feature of this lamp (Fig. 140) is the metal chimney *a* in the figure, which is secured in position by the gauze diaphragm. The chimney separates the intake air from the return and induces a brisk current through the lamp. These lamps are extinguished if tilted, because the spent air,

instead of going up the chimney, passes away on one side of the lamp, and, meeting the fresh air on its way to the flame, is carried back and puts out the light. Mueseler lamps may be either unbonneted, or they may be bonneted, in which case they are able to resist a high velocity.

Hepplewhite-Gray.—This lamp (Fig. 141) is much used by officials. The feed air enters the lamp at the top, through the tubes *a*, and passes to the flame through the gauze ring *b*; the products of combustion pass away through the upper gauze *c*, and out at the top of the lamp; the outlet is restricted at *d* in order to keep the upper portion of the lamp filled with the products of combustion. The lamp can be extinguished by means of a "shut-off," which is a contrivance for closing the inlets and so causing the lamp to be extinguished by cutting off the supply of air. The shut-off consists of a flat brass ring placed over the tops of the tubes (shown in section at *e*, Fig. 141). The ring is pierced with holes so arranged that they come exactly over the tops of the tubes and offer no obstruction. To extinguish the light, the ring is turned until the holes in it no longer coincide with the tops of the tubes, which are then covered with the solid portions of the ring. Shut-off arrangements can be applied to any bonneted lamps, but are more commonly arranged to close the outlet holes at the top of the bonnet.

The advantage gained by having the air inlet at the top of the lamp is that a very thin layer of gas can be detected. This lamp has a conical glass, which enables the roof to be inspected more easily, as the light is not obstructed by projecting bonnet and flange, as is the case in most lamps.

The Thorneburry.—This lamp is provided with two concentric glasses having a space between them; a metal chimney extends from the top of the inner glass, and is surrounded by a gauze and bonnet. The feed air enters through holes in the base of the bonnet, passes through a gauze and down the space between the glasses; it then goes through another gauze to the flame. The products of combustion pass up the metal chimney,

through the large gauze, and away through holes in the upper part of the bonnet.

Thorneburry lamps are rather heavy, but are safe and give a very good light. They are made in large sizes, to be hung in pass-byes and at other busy places.

Illuminants.—The oil burned in a safety-lamp may be either vegetable, animal, or mineral; frequently a mixture is employed.

Mineral oils give a good light, and do not require as much attention as the other varieties, being less liable to form crusts on the wick; they are, however, rather more dangerous to use, and none should be employed which have a flashing point of under 73 degrees Fahr. The wick tube should be flat, about $\frac{1}{2}$ inch in width, and the top of the tube should stand about $\frac{1}{2}$ inch above the bottom of the glass.

The wick should be short, and fit loosely in the tube, to allow the oil to circulate freely, to which end some wick tubes are made with a corrugation in one side. The height of the wick is regulated by a "pricker," which consists of a wire passing in a small tube through the oil-vessel, and bent at the end in such a manner as to enable it to catch the wick through a slot in the tube provided for the purpose. The light given by a safety-lamp under ordinary conditions varies between $\frac{1}{2}$ and $\frac{3}{4}$ candle power, though towards the end of the shift, and in a dusty or impure atmosphere, the light frequently falls below the former figure.

The cost for oil should not reach $\frac{1}{4}d.$ per lamp per shift, and the total cost, including cleaning, repairs, and renewals, should not exceed $\frac{3}{4}d.$ per lamp per shift.

Locks.—All safety-lamps must be securely locked before being taken into workings in which naked lights are prohibited.

The simplest form of a really efficient lock is the lead rivet; this consists of a hasp secured to the lamp frame and passed over a brass staple fixed to the lamp bottom. The hasp is secured by means of a small lead pin, which is pushed through the staple and firmly riveted, by means of a press, in such a manner that the hasp cannot be lifted without the lead pin

being cut. Additional security is attained by stamping some device upon the rivets, by changeable dies fitted on to the press.

Some lamps are locked by means of bolts closed with springs and withdrawn by magnets, by compressed air, or by a partial vacuum; this type of lock is very secure when in good order, but in some forms can be opened after the lamp has been subjected to the rough usage often met with in mines. Lamps of the Protector type are so arranged that the action of unscrewing the oil-vessel draws a sleeve over the flame and extinguishes it when the lamp is opened.

Relighting Lamps.—In many collieries no naked lights are allowed in the mine, and all lamps must be sent out to be relighted when they have become accidentally extinguished. This is a very serious inconvenience, as when a man, working in a remote district, loses his light, a couple of hours or more may elapse before he can get it back relighted.

Lamps are now made that can be lighted whilst locked; this is accomplished either by means of matches carried inside the lamps, or by the aid of electricity. The Wolff lamp is an example of the former class; the matches are arranged in a waxed tape, which is coiled spirally in a box within the lamp. These matches can be struck one at a time just over the wick tube by manipulating a lever. These lamps are very convenient for the use of officials who have to travel the workings and airways alone, but can hardly be recommended for the workmen, as the accident which extinguished the light may have damaged the lamp, and it should not be relighted until it has been carefully examined.

Lamps may be lit by electricity either by heating a platinum wire placed just over the wick tube by means of a low-tension current, or by a spark produced by a current of high tension. The former method is employed with lamps which burn the vapour given off from light oils, and the latter with the ordinary oils burned at wicks. The electric current is usually furnished by an accumulator, and the wire is heated or spark produced when a connection is made between the two terminals of the

battery and those on the lamp; the lamp itself usually forms one terminal, and an insulated wire passing through the lamp bottom serves as the other.

To light a lamp, it is placed on a stand which is connected to the battery and has two terminals, one of which presses against the insulated wire and the other against the lamp bottom. The current then passes through the circuit in the lamp, and either heats the wire or produces a spark, as the case may be. The relighting batteries should be locked, and placed in charge of a responsible person, who should examine every lamp before lighting it.

Electric Lamps.—Portable electric lamps have been introduced for use in mines, but their employment has not as yet become at all general.

They are more costly than ordinary safety-lamps, and, though they give more light, it is not so well diffused. So long as the glass remains intact, electric lamps cannot cause an explosion, but should the glass become broken, an explosive mixture might be ignited; in this they are only on an equality with the ordinary oil lamps.

The chief drawback to the use of electric lamps is that they give no indications of the presence of gas or "damp," and as the workman is almost entirely dependent upon the behaviour of his lamp as a guide to the state of the air, this is a rather serious matter.

One of the best-known electric lamps is the Sussman, which consists of a small incandescent lamp mounted on the top of a storage battery.

The lamp and battery measure $2\frac{3}{4}$ by $2\frac{3}{4}$ by 8 inches, and weigh from $3\frac{1}{2}$ to 4 lbs., which is about the weight of an ordinary oil lamp.

The battery is of the Faure or pasted type, and contains no liquid that can spill if overturned; two cells are employed, each having an E.M.F. of 2 volts. The cells are charged by a dynamo, the operation taking 12 or 13 hours; and when fully charged the light is maintained for 8 or 9 hours.

The advantages claimed for electric lamps are—

1. Absolute safety (unless the glass breaks).

2. They keep quite cool in any atmosphere, and do not go out if overturned.

3. They can be held at any angle in order to facilitate the examination of the roof.

4. They can be turned on or off at will, so that, in case a set of men should become imprisoned, a light could be maintained for a long time by using one lamp at once.

A few electric lamps should be kept at every colliery, as they are of great value to parties exploring a mine after an explosion has occurred, or for any similar work. They should, however, be used in conjunction with ordinary safety-lamps, so that the presence of poisonous or explosive gas may be detected.

Fire-damp Indicators.—The presence of gas (CH_4) in the atmosphere of a mine is detected by the behaviour of the flame of the safety-lamp.

To test for gas the lamp flame is drawn down very low, so that it may not dazzle the eyes of the observer; the lamp is then raised slowly and carefully into the place in which gas is expected and the appearance of the flame noted. When about $2\frac{1}{2}$ per cent. of gas is present the flame is slightly drawn, flickers a little, and shows a very faint blue cap; as the percentage of gas increases these effects become more and more marked. If less than $2\frac{1}{2}$ per cent. of gas is present, no visible effect is produced on the lamp flame, so that other means have to be adopted for measuring the percentage of gas present when it forms a proportion of less than about $\frac{1}{40}$ of the atmosphere.

It is desirable to measure very small proportions of gas, because the presence of a very small percentage greatly adds to the explosive properties of coal-dust; and by knowing what proportion of gas is present in the return airways under normal conditions, the effect of a reduction in the quantity of the air can be ascertained.

Instruments for measuring minute percentages of fire-damp are known as fire-damp indicators. Many different types have been introduced, depending for their action upon the following principles:—

1. The behaviour of certain classes of flame when burning in an atmosphere in which gas is present.

2. The contraction of the volume of a mixture of air and gas when in contact with a heated platinum wire, owing to the burning of the gas.

3. The greater amount of heat or light given off from a heated wire when the mixture exposed to the wire contains gas.

4. The difference in the rate of diffusion of gas and air.

5. The difference in the specific gravity of gas and air.

6. The difference in sound given off by a tuning-fork if gas is present.

Those designed upon the first of these principles are the most practical and successful.

The oldest of these is the *Pieler*, which consists of a large lamp burning pure alcohol. In construction the lamp is somewhat similar to a large Davy, but the wick is of the Argand type, that is, it is wrapped around a tube through which air passes to the flame. When $\frac{1}{4}$ per cent. of gas is present a cap of $1\frac{1}{8}$ inch in height is produced, and when the atmosphere contains 2 per cent. of gas the cap extends right to the top of the lamp.

In its original form this lamp is highly dangerous, even in a current of very moderate velocity.

Clowes' Hydrogen Fire-damp Detector.—This appliance is shown in Fig. 142. It consists of a detachable steel cylinder, *a*, fitted to an ordinary Hepplewhite-Gray lamp. This cylinder is about 5 inches long and 1 inch in diameter; it contains hydrogen at a pressure of about 100 atmospheres, and is attached to the lamp by means of a clip and by the screw *b*. A small tube, *c*, runs from the cylinder through the oil-vessel, and extends a little above the lamp-wick, the flow of the hydrogen being regulated by means of the screw *d*. A test is first made with the oil flame in the usual way, the hydrogen

being shut off. If 3 per cent. or more gas is present, it is detected by a cap on the oil flame, but if no cap is shown, the hydrogen is turned on and lights at the oil flame; this latter is then extinguished by drawing down the wick, and the test made again with the hydrogen flame. As this flame is extremely hot and non-luminous, a cap is shown when a very small percentage of gas is present; $\frac{1}{4}$ per cent. is said to give

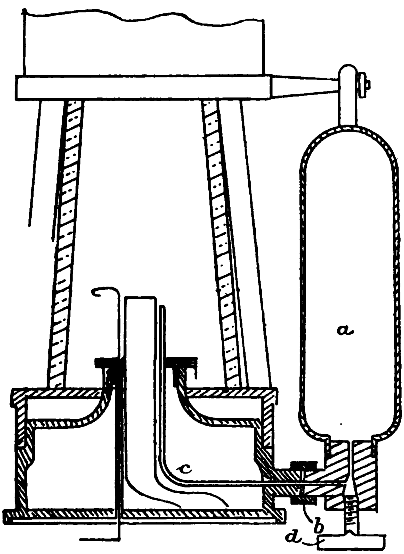


FIG. 142.—Clowes' fire-damp detector.

a cap about $\frac{1}{2}$ inch in height, and 2 per cent. gives a cap of about $1\frac{1}{2}$ inch.

After the test has been made, the wick is pushed up and lit at the hydrogen flame; the hydrogen is turned off and the cylinder detached.

Stokes' Fire-damp Indicator.—This arrangement is somewhat similar in principle and application to Dr. Clowes' apparatus, except that pure alcohol is employed instead of hydrogen.

The alcohol is contained in a small vessel fitted with a

long tube, and is used in conjunction with a safety-lamp which has a small pipe running vertically through the oil-vessel and closed with a spring cap. To make a test with the alcohol flame, the tube of the alcohol vessel is pushed through the pipe; the alcohol is then lit at the lamp flame, which is drawn down and extinguished.

The Beard-Mackie Gas Indicator.—This apparatus can be fitted to an ordinary safety-lamp; the principle upon which it depends is the absorption of gas by platinum wire, causing incandescence. It consists of a small frame of brass and platinum wires fixed vertically over the lamp flame. This frame is arranged like a ladder, the brass wires forming the supports, and the platinum wire, 6 staves. The height of the flame is regulated by a thin iron wire placed below the lowest platinum strand. The presence of gas causes the platinum wires to glow, and the number of wires affected indicates the percentage of gas present. If the lowest strand only glows, $\frac{1}{2}$ per cent. of gas is present; if the two lowest, 1 per cent.; and if all, 3 per cent.

CHAPTER XXIII.

WINDING.

Head Gear.—The pulleys over which the ropes pass from the winding drum to the cages are mounted upon a head-gear or pulley frame. Formerly head gears were always constructed of timber, but now they are more commonly built of steel, owing to the liability of timber frames to catch fire.

Fig. 143 shows a steel head gear 75 feet high, suitable for a 10 or 12 ton load. The main legs *a* are built up of four angle steel bars, each $3\frac{1}{2}$ by $3\frac{1}{2}$ by $\frac{1}{2}$ inch, braced together by flat bars of $2\frac{1}{2}$ by $\frac{1}{2}$ inch steel. The angles are fixed to form the corners of a square, each side of which measures 1 foot 9 inches. The front legs *b* are of $3\frac{1}{2}$ by $3\frac{1}{2}$ by $\frac{3}{8}$ inch steel angles, built up in a similar manner to the main legs. Both main and front legs are vertical in side elevation, but incline towards each other at the rate of about 1 in 9 in front elevation. The back-stays *c* are of similar strength and construction to the main legs, and the pillars against which they abut are prolongations of the engine-beds.

The plates for the detaching hooks are carried by the cross girders *d*; and the guide ropes are hung from the upper girders *e*. The pit-bank is not level with the foundations, but is a few feet above the girders *f*, which carry the props upon which the cage rests when the tubs are being changed.

The centres of the pulleys should rest upon the main legs.

Winding Pulleys.—These are made with cast-iron boss and rim connected by round rods of wrought iron. They are now made up to about 20 feet in diameter. The rims should be as

light as possible, consistent with strength and durability, as, if they are heavy, the pulleys may spin after the cage is at bank and the rope is stationary. Pulleys should run quite truly, otherwise much extra wear is put upon both ropes and pulleys. They are usually made in one piece, but to facilitate carriage they are sometimes made in halves or quarters and bolted together.

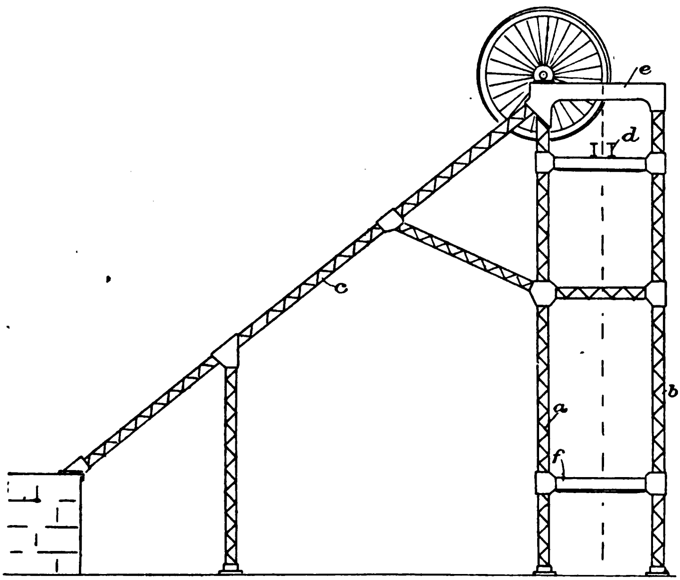


FIG. 143.—Steel head gear.

Cages.—The tubs are conveyed up and down the shaft in cages. These cages carry from one to twelve tubs, and have from one to six decks, varying according to the size of the shaft and tubs and the power of the engines. The weight of the cage is usually about 50 per cent. of the total weight of cage, corves, and coal.

There are usually two cages in each shaft, one of which descends whilst the other ascends, and as the ropes from both

cages are wound in the reverse direction round the same drum, the weight of the descending cage and corves balances the weight of the cage and corves which are being wound up the shaft. This leaves the winding engine only the weight of the coal and rope to deal with (unless the weight of the rope is balanced). In some of the old collieries in the North of England there are four cages in each shaft, each pair being

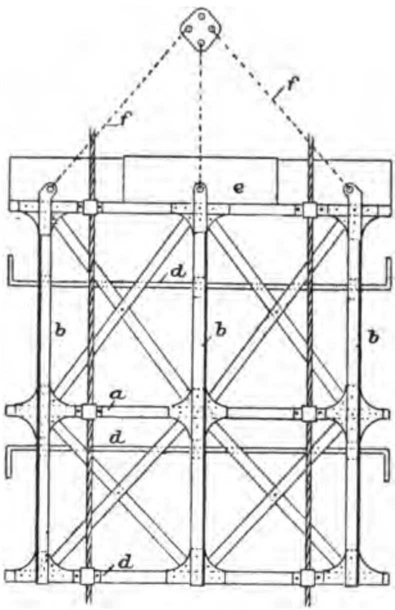


FIG. 144.—Double-decked cage.

worked by an independent engine. In other districts one engine is arranged to wind from two shafts, there being one cage in each.

Fig. 144 shows a double-decked cage suitable for carrying two corves on each deck. The horizontal rings *a*, *a*, which form the decks of the cage, are of angle steel, and are held in position by being riveted to the vertical angle-steel bars or

hangers *b*, *b*. The decks are provided with rails upon which the corves stand, the latter being kept in position by the bars *d*, which are raised when they are changed. A sheet-iron bonnet, *c*, is provided to protect the men travelling in the cage from anything which might fall down the shaft. The cage chains *f*, *f* are secured to wrought-iron plates riveted to the tops of the hangers. Each pair of chains is connected at the top by a ring, and each of the three rings is connected to a plate as shown in the figure.

In a few instances no cage chains are used, their place being taken by a strong bar, which passes through the bonnet

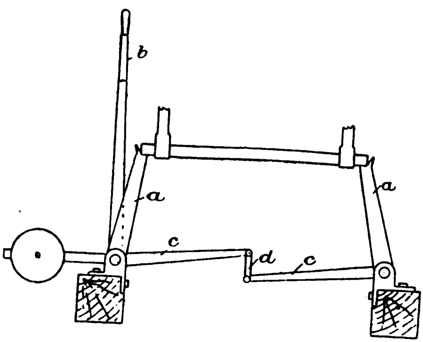


FIG. 145.—Cage props.

in the centre of the cage, and is secured to the cage frame through a coach-spring.

Cages should be as light and rigid as possible, and for this reason they are usually constructed of angle, tee, and channel steel; the joints between the vertical hangers and horizontal rings are often made by means of an intermediate plate, which gives a larger area for the rivets and stiffens the whole frame.

Props or Keps.—When the cage reaches the surface, the full corves are pushed off, and empty corves pushed on in their place. Whilst this is being done, the cage rests on props or keps, a set of which is shown in Fig. 145. As the cage is

drawn up, it pushes the bars *a*, *a* back, and they fall forward again and bar the way as soon as the cage is drawn above them. The engine-man then lowers the cage on to the props, which hold it whilst the corves are being changed. After this has been done the cage is raised clear of the props, and the banksman draws them out by means of the lever *b*, holding them back until the cage has passed. There are four props to hold each cage, set in pairs, each pair being keyed to one shaft. The rods *c*, *c* and link *d* connect the pairs of props as shown in the figure in such a manner that all fall to or are pushed back simultaneously.

It will be noticed that before the cage can be lowered it has to be raised clear of the props to enable them to be withdrawn. To avoid the delay occasioned by this, a form of props has been introduced which can be withdrawn whilst the weight of the cage is upon them.

Props of this type decrease the time occupied in changing the corves, but have to be used with care, and no slack rope allowed.

Conductors.—Winding cages run on guides or conductors in order to keep them from catching the sides of the shaft or colliding with each other. There are three kinds of guides in general use, namely, timber, rail, and rope guides.

Timber conductors are now rarely fixed in important shafts, as they split and rot and require considerable attention. They are, however, in use at many of the older collieries. They consist of straight pine rods, varying in section between 4 by 3 inches and 6 by 4 inches. They are fixed vertically down the shaft by being bolted to buntons spaced about 6 feet apart. Buntons are timbers placed across the shaft and secured by being let into the sides. The cages are fitted with cast-iron shoes, which work loosely on the guide and prevent oscillation.

Two guides are usually provided for each cage, one being placed at either end, two rows of buntons only being necessary. As guides fixed in this way block the entrances to the cage, they are broken off at the top and bottom of the shaft, and their place

taken by short guides or off-take rods, which are placed on either side of the cage and not at the ends.

Rail guides are usually employed in deep shafts of limited area, where there is not sufficient clearance between cages and sides to admit of the use of rope guides. Flat-bottomed steel rails, weighing 50 or 60 lbs. per yard, are usually employed; they are bolted to buntons or girders placed about 10 feet apart, the general arrangement being similar to that adopted with timber guides. Rail guides are very durable and efficient, but are expensive in first cost, and are somewhat noisy in use.

Wire-rope guides are largely used, and are preferable to any form of rigid conductors under ordinary conditions. The ropes are suspended from the head-gear, and hang down the shaft to the sump, where they are weighted with cast-iron weights to keep them taut.

The ropes themselves are similar in construction to ordinary round wire ropes, except that each strand consists of one solid wire of large diameter, instead of being built up of thin wires. This is to enable the ropes to stand great wear without the wires breaking. Each rope is hung from the head gear by means of several pairs of strong clamps. By this arrangement a length of spare rope can be provided, which is desirable, because the end of the rope which is in the sump is apt to corrode. Each rope is weighted in the sump by cast-iron weights, about 1 ton being allowed for every 200 yards in length of rope. The ropes are passed through staples fixed to timbers in the sump to limit the oscillation.

Four guides are usually provided for each cage, the cages being fitted with small cast-iron thimbles, which embrace the guides. Loose guides are often hung down the shafts between the cages in addition to the other guides, or sometimes instead of the inner ones.

Fig. 146 is the plan of a shaft fitted with these loose or "rubber" guides. The cages are brought close together and kept from contact by the loose guides *a*, *a*, which are not in any way attached to the cages. By using these loose guides and bringing the cages close together, more room is left

between the cage corners and shaft sides. There is usually considerable oscillation at the meeting of the cages, owing to the restriction of the area for the passage of the air.

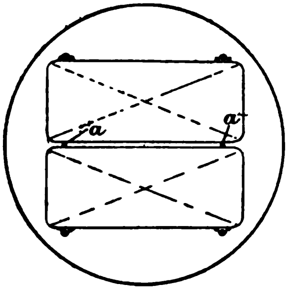


FIG. 146.—Arrangements of wire rope guides.

Ropes and Chains.—Winding ropes are now always made of steel. They may be either round or flat. Flat ropes are never employed at new collieries, owing to their greater weight and cost. Round ropes are made up of strands twisted together, each strand being composed of thin wires; usually the strands are coiled round a central core of hemp. Flat ropes are made up of several thin round ropes placed side by side and stitched together with wire. Several qualities of steel are used in the manufacture of colliery ropes. Plough steel has the highest tensile strength, 110 to 120 tons per square inch, and Bessemer or mild steel the lowest, 40 to 45 tons per square inch. Round winding or hauling ropes may be either of *Ordinary Lay*, *Lang's Lay*, or *Locked Coil.*

In ropes of ordinary lay the wires in each strand are twisted in the *opposite* direction to the strands in the rope, as

FIG. 147.—Steel rope—ordinary lay.

shown in Fig. 147. This method of twisting the rope results in the crowns of the strands being exposed to the greatest amount of wear, and leads to the wires breaking at those points.

Fig. 148 shows a rope constructed on Lang's lay principle;

in this case the wires which form the strands are twisted in the *same* direction as the strands themselves. By this arrangement a much larger surface is exposed to friction, which results

FIG. 148.—Steel rope--Lang's lay.

in the wear being more uniform, hence the ropes have a longer life.

Fig. 149 shows a locked-coil rope, as constructed by Messrs. George Elliott & Co., of London and Cardiff. The inner strand is composed of ordinary round wires, but the outer coils are of wires of special section, coiled spirally in such a manner as to interlock and form a perfectly smooth working surface.

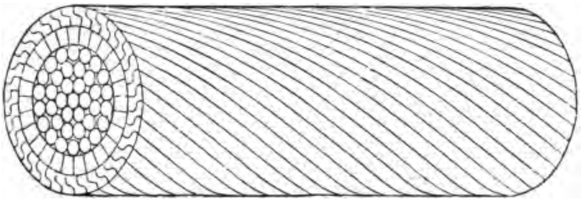


FIG. 149.—Locked-coil rope.

The advantages of locked-coil ropes are—

1. They do not twist in working; this is frequently of importance, especially in sinking.

2. They are very flexible, and will therefore work round small drums and pulleys.

3. They possess a large and uniform working surface.

4. They are of less weight and size, strength for strength, than ropes of ordinary construction.

The number of wires of which a rope is composed depends

upon the circumstances under which it will have to work. If a rope has to pass round a small pulley or sharp bends, the wires must be of small diameter, and their number increased in order to give the necessary flexibility.

The approximate weight of round ropes may be calculated by the following formulæ:—

C = circumference in inches; W = weight in lbs. per yard.

$$\text{Hemp ropes, } W = \frac{C^2}{8}$$

$$\text{Iron or steel ropes, } W = \frac{C^2}{2}$$

For example, a steel rope 5 inches in circumference and 500 yards long wouid weigh—

$$\frac{5 \times 5 \times 500}{2} = 6250 \text{ lbs.} = 55{\cdot}8 \text{ cwts.}$$

The following are the formulæ for determining the approximate breaking strains of round ropes:—

B = breaking strain in tons; C = circumference of rope in inches.

Hemp ...	$B = C^2 \times 0{\cdot}3$, and	$C = \sqrt{\frac{B}{0.3}}$
Iron ...	$B = B^2 \times 1{\cdot}5$, „	$C = \sqrt{\frac{B}{1{\cdot}5}}$
Mild steel	$B = C^2 \times 2$, „	$C = \sqrt{\frac{B}{2}}$
Plough steel	$B = C^2 \times 4$, „	$C = \sqrt{\frac{B}{4}}$

The factor of safety for winding ropes is usually taken at 10. This means that the working load of a winding rope should be only $\frac{1}{10}$ of the breaking strain.

The reasons for employing so high a factor of safety are—

1. If the chains are allowed to become slack, the strain on the rope may greatly exceed the normal load.

2. The capple is rarely as strong as the rope.

3. The wires harden with use and the strength of the rope decreases.

Examples.—(1) Find the safe working load of a plough-steel rope $4\frac{1}{2}$ inches in circumference.

$B = 4{\cdot}5 \times 4{\cdot}5 \times 4 = 81$ tons; taking 10 as the factor of safety, the safe load is $\frac{81}{10} = 8{\cdot}1$ tons.

(2) What size mild-steel rope would be required for a working load of 4 tons?

The rope must have a breaking strain of $4 \times 10 = 40$ tons, and the circumference is $C = \sqrt{\frac{40}{2}} = 4{\cdot}47$ inches.

In estimating the load upon a winding rope the weight of the rope itself must be taken into account. In deep pits the weight of the rope forms a very considerable proportion of the load.

Example.—Find size of plough-steel rope to lift a load of 4 tons from a depth of 500 yards.

Using the formulæ given above, and taking the factor of safety as 10—

$$4C^2 = 10\left(\frac{4 + \frac{C^2}{2} \times 500}{2240}\right)$$

$$\begin{aligned} 4C^2 &= 40 + \tfrac{250}{224}C^2 \\ 896C^2 &= 2960 + 250C^2 \\ 646C^2 &= 8960 \\ C^2 &= 14, \text{ and } C = 3{\cdot}7 \text{ inches.} \end{aligned}$$

Rope Cappings.—Rope ends are fitted with cappings or sockets. Fig. 150 shows three views of a socket for a round winding rope.

To fit a socket of this description, the rope, where it comes into it, is wrapped round with copper wire, the wrapping extending to within a foot or two of the rope end. A hollow cone is then slipped over the wrapping, and the loose wires which extend beyond are bent back over it, as shown in

section at B and outside elevation at C (Fig. 150). The end of the socket is then hammered firmly into position, and the three hoops, which have previously been slipped up the rope, are driven down on to the socket.

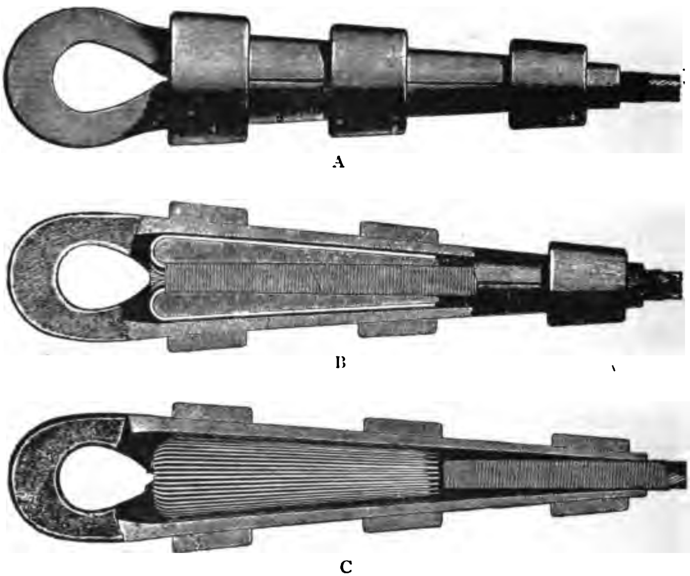


FIG. 150.—Socket for round ropes.

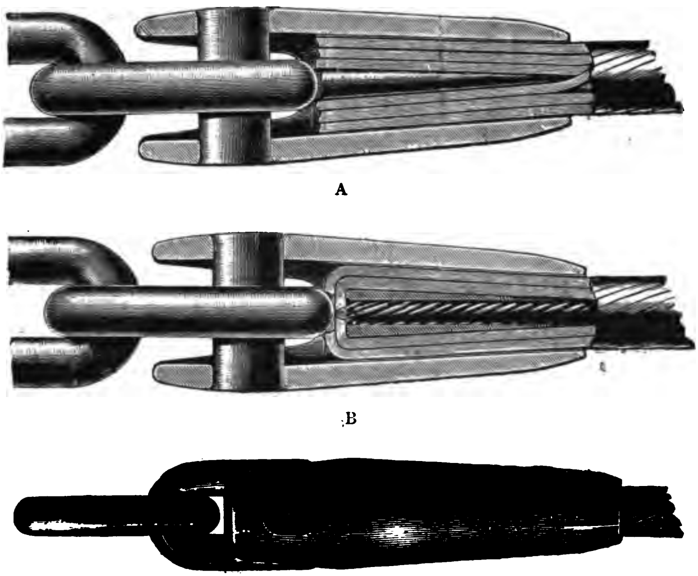
Locked-coil ropes require a special form of socket. The examples shown in Fig. 151 are supplied by Messrs. George Elliott & Co.

These cappings consist of an open-ended conical socket, furnished with a bolt to carry the cage chains. To fix this, it is first slipped down the rope, and the rope end enlarged and made wedge-shaped, either by driving into the rope a copper or soft iron conical plug, as at A; or by means of half-round wedges, as at B (Fig. 151). The socket is then drawn down over the wedge-shaped end, and the bolt which has to carry the link inserted.

Chains.—The approximate safe working load of chains may be calculated by the following formula :—

S = working load in tons ; D = diameter in eighths of an inch.

$$S = \frac{D^2}{10}, \text{ and } D = \sqrt{S \times 10}$$



A

B

C

FIG. 151.—Sockets for locked-coil ropes.

Examples.—(1) Find the safe working load of a chain made of $\frac{3}{4}$-inch iron.

$$S = \frac{6 \times 6}{10} = 3\cdot6 \text{ tons}$$

(2) What size chain would be required for a working load of 10 tons?

$$D = \sqrt{10 \times 10} = \frac{10}{8} \text{ inch} = 1\tfrac{1}{4} \text{ inch}$$

Winding Engines.—The work that winding engines have to perform is very different from that required of most engines. They are continually stopping and starting, and both load and speed vary at almost every stroke. For these reasons, winding engines are usually built with a view to strength, simplicity, and handiness, rather than to economy. The great majority of winding engines are simple, non-condensing engines, and many work with but little expansion. The tendency now is for collieries to use steam at much higher pressure than formerly, and winding engines are being built to take advantage of this by working compound. The steam from winding engines is condensed at a few places by independent condensers.

Nearly all the large winding engines which are now built are fitted with some form of automatic cut-off gear, which comes into action as soon as the engines have attained their maximum speed, and cuts off the steam as early in each stroke as is necessary to maintain that speed.

Winding engines may be either vertical or horizontal, but the latter are by far the more common. Except for very small places, winding engines are direct-acting, that is, the connecting rods are coupled direct to cranks on the drum shaft without the intervention of gearing. A pair of coupled engines should be employed, their cranks being at right angles to each other; this ensures smooth running, and one engine s exerting its maximum power whilst the other is on its dead-centre. Double-beat equilibrium valves are usually employed, being much more easy for the engine-man to handle than the ordinary slide-valve. Fig. 152 shows a large pair of winding engines built by Messrs. Thornewell & Warham, of Burton-on-Trent.

Equalizing the Load on Winding Engines.—The load upon a winding engine is made up of the weight of unbalanced rope plus the weight of the coal: the weight of the ascending cage and corves being balanced by that of the descending ones. The length of rope which has to be raised becomes shorter as the cage ascends, and part of it is balanced by the

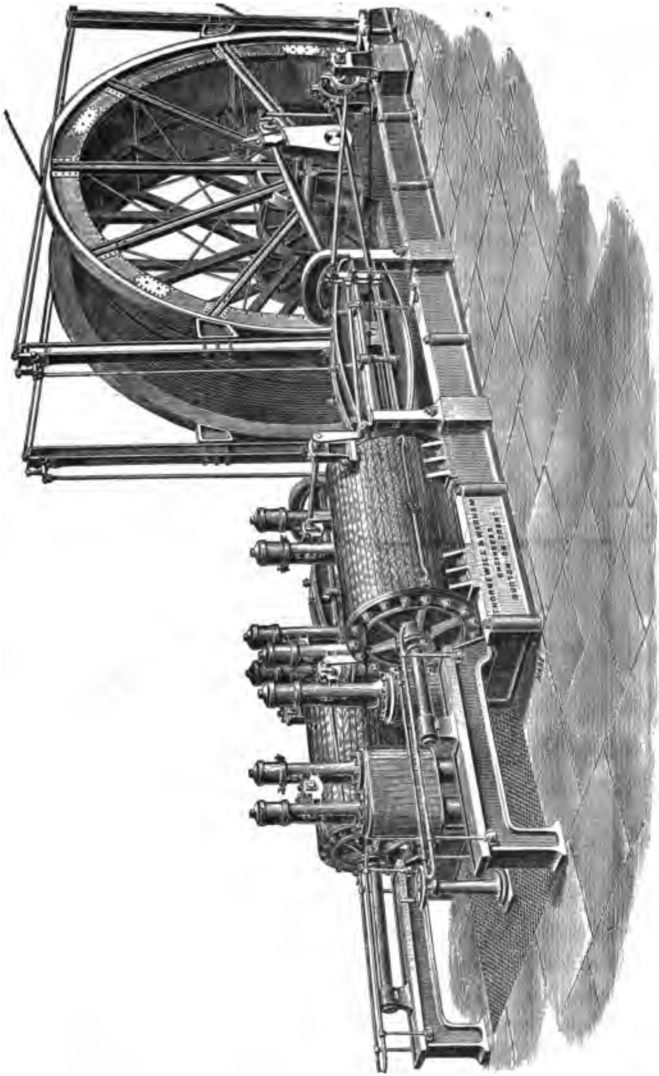


FIG. 152.—Horizontal winding engines.

descending rope. When the cages meet in the centre of the shaft, the descending ropeis equal in length to the ascending rope, so that the weight of the ropes is balanced and has no influence on the engines. When the full cage is at the pit-top, the whole weight of rope is on the empty side and is assisting the engines, so that if the weight of the rope were just equal to the weight of the coal, there would be no strain on the engine at all. If the rope were heavier than the coal, the descending would overbalance the ascending load, giving rise to what is known as a negative load.

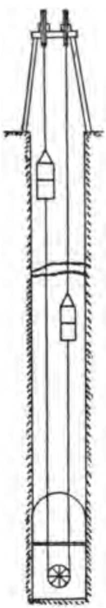


FIG. 153.—Balance rope.

For example, if the coal raised per wind weighed 3000 lbs., and the weights of the ropes were 5000 lbs. each, at the commencement of the wind there would be $5000 + 3000 = 8000$ lbs. against the engines; in the middle the load would be 3000 lbs. only, as the ropes would be balanced; and at the end of the wind the load would be $3000 - 5000 = -2000$, that is, a negative load of 2000 lbs.

So that, although the average load is 3000 lbs., the engines have to raise 8000 lbs. at the beginning of the wind, and hold back 2000 lbs. at the end. This is, of course, very detrimental, as power has to be provided for raising nearly three times the average load, and engines working under such conditions cannot well be economical.

Balance Ropes.—The simplest method of equalizing the weight upon a winding engine is by means of a balance or tail rope. The general arrangement of a balance rope is shown in Fig. 153. A rope, equal in weight to each of the winding ropes, is hung from the bottom of one cage to the bottom of the other; it makes a turn in the sump, where it may either pass round a pulley, as shown in the figure, or simply hang in a loop. With this arrangement the weight of balance rope plus weight of winding rope is

always equal for both cages. These balance ropes are very common; they save steam, and shorten the time spent in winding; the only objection to them is that they put extra strain upon the capping of the winding ropes, which is their weakest point.

Spiral Drums.—The weight upon a winding engine may be equalized by the employment of a spiral drum, the diameter of the drum at the commencement of the wind being less than the diameter at the end; so that at the beginning of the wind the engine is raising the full cage and rope through a shorter lever than that from which the empty cage, the weight of which assists the engine, is hung.

The construction of a spiral drum is shown in Fig. 154.

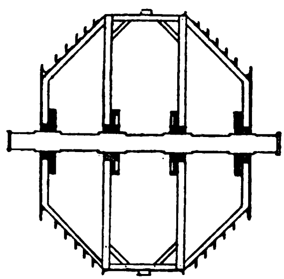


FIG. 154.—Spiral drum.

The framework of the drum is constructed of channel steel; a spiral groove runs round the inclined portion, and gradually mounts to the parallel portion at the top, which is plated with sheet-iron in the usual way.

The load on the engine at the commencement of the wind is: weight of full cage and rope multiplied by small diameter of drum, minus weight of empty cage multiplied by the large diameter. At the end of the wind the load is: weight of full cage multiplied by large diameter of drum, minus weight of empty cage and rope multiplied by the small diameter. To get a perfect balance, the load at the commencement must equal the load at the end of the wind.

From this we get the following formulæ for finding the relative diameters of spiral drums :—

$$L = \frac{S(2R + F + E)}{F + E}, \text{ and } S = \frac{L(E + F)}{2R + F + E}$$

L = largest diameter of drum ;
S = smallest " "
R = weight of rope ;
F = " full cage ;
E = " empty cage.

Example.—Rope weighs 5000 lbs., full cage 10,000 lbs., empty cage 7000 lbs., smallest diameter of drum 12 feet : find largest diameter.

$$L = \frac{12(10000 + 10000 + 7000)}{10000 + 7000} = 19{\cdot}06 \text{ feet}$$

It is customary to overbalance the weight of the rope by making the largest diameter greater than that given by calculations. This enables the engines to start very quickly, and assists in pulling them up at the end of the wind ; in other words, it tends to counteract the inertia of the drum. The objection to spiral drums is their great weight, width, and cost.

Chain and Staple-pit.—This method of equalizing the load on a winding engine is never applied with modern plants, but is in use with many old engines, chiefly in the North of England. A small drum is keyed on to the main drum shaft, and upon it is coiled a flat chain, to the end of which a bunch of heavy chains is attached and hangs down a staple-pit. When the winding engine begins to lift its load, it is assisted by the weight of the heavy chains hanging in the staple. As the load is raised, the chains are coiled off the drum, and lie at the bottom of the staple. By the time the cages meet in the shaft, the whole of the light chain is coiled off the drum, and all the heavy chains lie in the bottom of the staple. During the remainder of the wind the chains are raised again, the light chain being coiled on the drum in the reverse direction. In this

manner the weight of the chains assists the engine at the commencement of the wind when the load is greatest, and retards it at the end when the load is lightest.

Cage Indicators.—Every winding engine must be provided with an indicator to enable the engine-man to know the exact position of the cages in the shaft. The simplest form of indicator consists of a small drum driven from the main shaft of the engine, upon which is coiled a light chain. The chain is carried over a small pulley, and to it is attached a weight which slides in a vertical frame. As the cage is raised, the weight descends, and reaches the bottom of its run when the cages meet; the chain then coils on to the drum in the reverse direction, and raises the weight to the top of the frame as the cage reaches the top of the shaft. When the cage is nearing the surface the indicator is arranged to ring a bell which is attached to its frame.

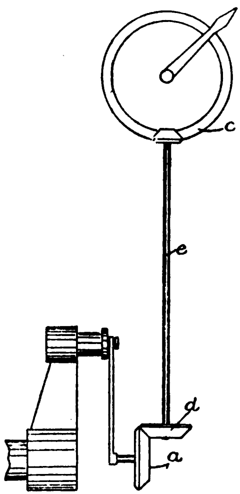


FIG. 155.—Cage indicator.

Fig. 155 shows a more modern and superior form of cage indicator. The bevel wheel *a* is driven by a light crank from the engine crank-pin, and drives the wheel *c* through the small wheels *a* and shaft *e*. *c* carries a pointer, which revolves round a dial and shows the position of the cages. The pointer moves backwards and forwards round the dial, making a complete revolution for each wind. Some indicators have two pointers, one making a complete revolution per stroke.

Brakes.—Every winding engine must be provided with a brake, which should be of sufficient power to hold the load if the engines should be disabled. Brakes depend upon friction for their action, and consist of a block or strap which is pressed against a brake ring or the drum. Brakes may be worked

either by steam or by foot; steam brakes are usually fitted to large engines.

The laws governing the friction of solids (Chapter XVII.) show that brakes do not require very large rubbing surfaces, and that the weight they hold depends upon the pressure between the brake and ring.

A brake suitable for a large pair of winding engines is shown in Fig. 156.

The brake ring *a* is of iron, turned to a true circle. The brake blocks *b*, *b* are of oak, and are bolted to the girders *c*, *c*, which are connected at their tops by the rod *d*. When the foot-plate is pressed down, the levers to which it is connected

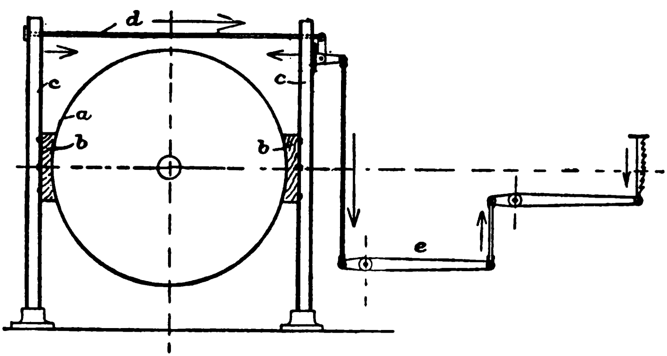


FIG. 156.—Brake for winding engines.

move in the direction of the arrows, and the brake blocks are forced towards each other, putting pressure upon the brake ring from opposite directions. Arrangements must be made for adjusting the levers as the blocks wear, by means of screws. If it is desired to actuate the brake by steam, the end of the lever *e* can be connected to the piston-rod of a small vertical cylinder.

Detaching Hooks.—If the engine-man fails to stop the engines when the cage reaches the surface, the ascending cage is pulled into the head-gear, and the descending cage is dashed into the pit bottom. This is known as an overwind. Detaching hooks

are designed to save the ascending cage from the effects of an overwind by liberating the rope and suspending the cage in the head-gear.

Fig. 157 shows King's detaching hook as seen when suspending a cage. It consists of four wrought-iron plates. The cage is hung from the outer plates *a* in the figure, and the rope is held by jaws on the inner plates *b*. The inner plates are pivoted on the central pin *c*. When the hook is carrying the load in the shaft, the wings *d*, *d* on the inner plates project beyond

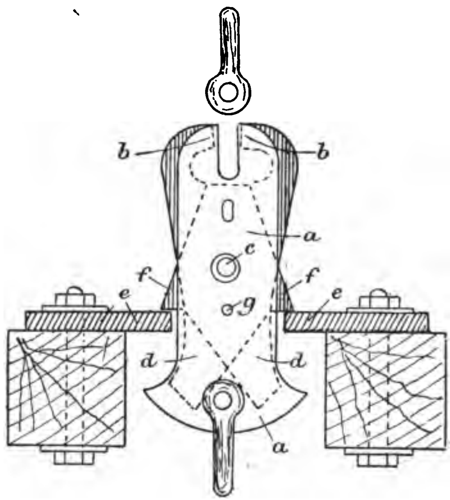


FIG. 157.—King's detaching hook.

the outer plates, but that portion of the plates shown shaded in the figure is flush. *e* is a strong iron plate bolted on to the head-gear, and through which the winding rope passes on the way from the pulley to the cage. If an overwind occurs, the hook is pulled into this plate, and the projecting wings *d*, *d* are knocked into the position shown in the figure. As the wings are forced inwards, the projections *f*, *f* are forced outwards, and hold the hook securely as in the figure; at the same time the jaws *b*, *b* are forced open and the winding rope liberated. The inner

plates are kept in position during the winding by means of the soft-iron rivet *g*, which passes through all four plates, and is sheared when the hook is pulled into the ring.

Winding from Two Levels.—When two seams have to be wound from one shaft, they should, if possible, be connected by a drift, and both wound from one level. If this cannot be arranged, the winding engine is fitted with a drum having two diameters, the larger diameter serving the deeper seam. This arrangement is objectionable for several reasons: the output from each seam is limited, and if more coal is turned from one seam than from the other, one cage has to travel the shaft empty, and the cost of handling the coal is increased.

When winding from two levels, the relative diameters of the drum vary with the depth of the two seams. Thus, if the depths to the seams were 360 and 410 yards respectively, and the smaller diameter of the drum were 12 feet, the larger diameter would be $\frac{410 \times 12}{360} = 13$ feet 8 inches.

Calculating the Size of Winding Engines.—The work done by a winding engine is spent on raising the load and on imparting velocity to it. The simplest method of finding the approximate size of winding engines for a given amount of work is to calculate the size of engine necessary to raise the load plus 25 per cent. for friction, and take this as one of a pair. Although this rule is largely employed and gives fairly accurate results, it is not scientifically correct, as the power spent in the acceleration of the moving mass is not taken into consideration.

In making calculations of this description, there are always several factors to assume, and to do this correctly requires a certain amount of judgment.

The piston speed of winding engines is usually from 400 to 600 feet per minute, and the cage speed from 25 to 50 feet per second; the larger the engines, the higher the piston and cage speed. The stroke is usually about twice the diameter of the cylinder, and the diameter of the drum about three or four times the length of stroke.

Example.—Find the size of a pair of winding engines to pull 1500 tons in 10 hours from a depth of 500 yards, the weight of the ropes being balanced, and the average pressure of steam on the piston 45 lbs. per square inch.

Tons in 10 hours, 1500; tons per hour, $\frac{1500}{10} = 150$.

Taking the cage speed at 35 feet per second, the number of seconds spent in a journey are $\frac{500 \times 3}{35} = 43$ nearly.

Allowing 12 seconds for changing corves, the time spent per wind is $43 + 12 = 55$ seconds.

Number of winds per hour $\frac{60 \times 60}{55} = 65{\cdot}5$.

Making an allowance of about 10 per cent. for minor delays, the number of journeys per hour may be taken at 60.

Tons per journey $\frac{150}{60} = 2\frac{1}{2} = 5600$ lbs.

If the diameter of the drum is four times the length of stroke, the leverage is $\frac{4 \times 3{\cdot}141}{2}$ to $1 = 6{\cdot}282$ to 1; that is, the load moves through 6·282 feet for every foot moved through by the piston upon which the pressure is applied; hence the pressure must be 6·282 times the load. The total pressure on piston, then, is $5600 \times 6{\cdot}282$, but 25 per cent. must be allowed for frictional resistances, so that the total pressure must be $\frac{5600 \times 6{\cdot}282 \times 125}{100}$. Taking the average steam pressure on piston at 45 lbs. per square inch, the area is $\frac{5600 \times 6{\cdot}282 \times 125}{45 \times 100}$, and the diameter $\sqrt{\frac{5600 \times 6{\cdot}282 \times 125}{45 \times 0{\cdot}7854 \times 100}}$ $= 35\frac{1}{4}$ **inches.**

A pair of engines would be required having 36-inch cylinders by 6-feet stroke, the drum being 24 feet in diameter.

If the weight of the rope were unbalanced, it would have to be added to the weight of the coal, and would necessitate larger engines.

CHAPTER XXIV.

HAULAGE.

THE conveyance of the coal from the workings to the shaft is usually performed in at least two operations. First, the coal has to be collected from the various working places, and taken to the pass-byes or sidings, from whence it is conveyed by the main haulage to the shafts. The pass-byes should be kept as near to the workings as possible, in order to reduce the length of the secondary haulage, which is much more costly than the main haulage.

Corves.—The vehicles in which the coal is carried are variously known as "corves," "tubs," "boxes," "trams," etc. They vary in capacity from about $3\frac{1}{2}$ cwts. to about 50 cwts. The average weight carried in the Midlands is from 9 to 12 cwts.; in Wales very much larger weights are commonly dealt with, even in thin seams.

Corves may be constructed either of wood or of iron; a wooden corf of ordinary construction, suitable for a load of 10 to 12 cwts., is shown in Fig. 158.

The body is rectangular in shape, and built of elm or larch boards $1\frac{1}{8}$ inch in thickness; the corners may be protected by external plates of $\frac{1}{8}$-inch sheet iron. The framework upon which the body is carried consists of two oak legs, *a*, *a*, each 6 by 3 inches, held in position by cross-pieces or spendrils. The body is secured to the framework by bolts, and by the bars *b*, *b*, which are made of 2 by $\frac{8}{4}$ inch flat wrought iron. These bars extend along the sides and across the body of the corf, stiffening it and bracing the whole

together. The legs are a few inches longer than the body, to form buffers, and are protected by a hoop-iron ring.

The draw-bar is made of 3 by $\frac{3}{4}$ inch best wrought iron. Various forms of couplings are employed. They may be either loose—in which case a hole is pierced in either end of the draw-bar, and the coupling provided with a couple of hooks—or one end of the draw-bar may be fitted with a link, and the other end with a hook. In the figure both link and hook are fitted at each end; this makes a secure coupling, and both ends of the corf are similar. The *wheels* and *axles* are of cast steel, the wheels being 12 inches and the axles $1\frac{1}{2}$ inch in diameter; the gauge of the road is 24 inches. Wheels may

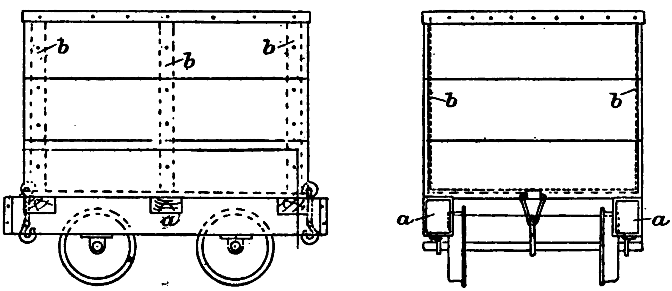


FIG. 158.—Timber corf.

be either fast on the axle and revolving with it, or loose; in the latter case, both wheels and axles are free to revolve. Wheels are now generally made fast on the axles, as they can be more efficiently lubricated, and are less liable to get out of order. Loose wheels are better adapted for running round very sharp curves, because the outer rail of a curve is longer than the inner rail, consequently, if both wheels are fast on the axle, one has to skid through a short distance. The wheel base is the distance between the centres of the axles; when the wheel base is short, the corves can be more easily handled and lifted on the road if they become derailed. A short wheel base causes the corf to be unstable and more easily upset when travelling steep roads.

The Road.—The corves run on rails. These are usually flat-bottomed, weighing from 15 to 30 lbs. per yard, and made in lengths of from 6 to 12 feet. They are spiked on to sleepers placed transversely, from 3 to 6 feet apart.

In the main roads, rails of heavy section are employed; they should be in long lengths, and the ends connected by fish-plates.

The rails in the working places are of lighter section and shorter lengths; they are spiked to sleepers, but no fish-plates are employed. Junctions are made either with short points, or, in the case of unimportant roads, with metal plates or hard-wood boards, the latter being preferred in steep seams.

When main roads change their direction, the curve connecting the straight lengths should be the arc of a circle tangential to both straight roads. The radius of the curve should be as large as possible, especially where the haulage is rapid. The proper method of laying out a curve is shown in Fig. 159. BA and CA are the straight roads which it is desired to connect by a tangential arc. Bisect the angle BAC by the line DA, draw GF parallel to CA at a distance from it equal to the radius of the required arc, cutting the line DA at G. G is then the centre from which the tangential arc must be struck.

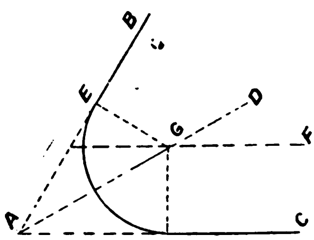


FIG. 159.—Method of setting out curve.

When corves run round a curve by gravity, they have a tendency to jump the outer rail, but when they are pulled round with a rope, they are liable to be pulled into the side, over the inner rail. To guard against the former contingency, the outer may be raised above the inner rail; and to prevent the latter, rubbing boards may be set at the inner side of the bend, against which the corves rub, and by which they are prevented from being pulled into the side.

Resistance to Traction.—In moving a corf along a level road, the resistance due to friction only has to be overcome. Frictional resistance is caused by the rubbing of the axles against the bearings, and the wheel flanges against the rails, and also by the inequalities in the road. The amount of friction depends very greatly upon the condition of the corves and road, and upon the efficiency of the lubrication.

Under ordinary conditions, the frictional resistance of a corf is about $\frac{1}{56}$ of its weight; this is called the coefficient of friction. The coefficient of friction may be experimentally determined in several ways. If a corf is pulled along a perfectly level road, the force required to keep it in motion may be found by a spring balance, and will be the measure of the frictional resistance offered by the corf; this, divided by the weight of the corf, gives the coefficient of friction. Thus, if the corf weighed 15 cwts., and required a force of 35 lbs. to keep it moving at a uniform velocity on a level road, the coefficient of friction would be $\frac{35}{15 \times 112} = \frac{1}{48}$.

The coefficient of friction can also be determined by allowing a corf to run down one incline and up another. If there were no friction, the momentum acquired in running down the one hill would carry the corf to exactly the same level up the other hill; owing to friction, there is always a loss in level, and this loss depends upon the coefficient of friction.

Where h = difference in level between the points where the corf stops and starts;

l = length of road traversed by the corf—

$$\frac{h}{l} = \text{coefficient of friction}$$

Example.—A corf runs 210 yards down one hill and 180 yards up another, the vertical distance between the starting and stopping points being $7\frac{1}{2}$ yards: what is the coefficient of friction?

$$\frac{7 \cdot 5}{390} = \frac{1}{52}$$

Hence the tractive force necessary to move the corf on a level road is $\frac{1}{52}$ of the weight.

If the plane along which the corf has to be moved is inclined, the force of gravity must be taken into account as well as the resistance due to friction.

Example.—A road dips at the rate of 1 in 3: find tractive force necessary to draw up a train of corves weighing 10 tons.

The dip is 1 vertical to 3 horizontal, hence the length, measured along the slope for every foot of vertical rise or fall, is $\sqrt{1^2 + 3^2} = \sqrt{10} = 3{\cdot}162$.

$$\text{Resistance due to gravity} = \frac{22400}{3{\cdot}162} = 7084 \text{ lbs.}$$

$$\text{,, \quad ,, \quad friction} = \frac{22400}{56} = 400 \text{ ,,}$$

$$\text{Total resistance} \ \ldots \quad \ldots \quad \ldots \quad 7484 \text{ ,,}$$

(See also Chapter XVII.)

The useful H.P. developed, if the speed is 8 miles per hour, is found as follows :—

$$\text{8 miles per hour} = \frac{8 \times 1760 \times 3}{60} = 704 \text{ feet per minute}$$

$$\text{foot-lbs. per minute} = 704 \times 7484$$

$$\text{and horse-power} = \frac{704 \times 7484}{33000} = 159{\cdot}6$$

Tramming.—When the distances are very short, or the road too low to admit ponies, the corves may be pushed by hand. The "trammer" or "putter" holds the corf with both hands, and pushes it along with the assistance of his head. Tramming is very costly, and every care should be taken to keep down the length of the roads along which corves must be trammed. In some methods of working thin seams, no ponies are employed, all the coal being trammed from the faces to the ropes, the extra expense of the tramming being compensated by the lessened cost of making the roadways.

The cost of tramming is enormously increased if the load

has to be pushed up an incline, so that tramming should never be allowed except on favourable gradients.

If the road is perfectly level, more force is necessary to move the full than the empty corves. For example, if the full corf weighs 10 cwts. and empty corf 3 cwts., the coefficient of friction being $\frac{1}{56}$, the force necessary to move a full corf on a level road is $\frac{10 \times 112}{56} = 20$ lbs., whilst the empty corf would only require a force of $\frac{3 \times 112}{56} = 6$ lbs. From this it follows that, in order to get the load exactly equal in both directions, the road must have a slight dip in favour of the full load.

The most favourable gradient may be found by the following formula:—

H = height of plane; L = length of plane; F = weight of full corf; E = weight of empty corf; K = coefficient of friction.

$$\frac{H}{L} = \frac{(F - E)K}{F + E}$$

Example.—Full corf weighs 15 cwts., and empty corf 5 cwts., and the coefficient of friction is $\frac{1}{56}$: find the gradient upon which the load is equal both ways.

$$\frac{H}{L} = \frac{(15 - 5)\frac{1}{56}}{15 + 5} = \frac{\frac{10}{56}}{20} = \frac{10}{56} \times \frac{1}{20} = \frac{1}{112}$$

Hence the desired gradient is 1 in 112 in favour of the load.

Horse Haulage.—Horses are employed to collect the coal from the various workings and haul it to the sidings. Where gradients are favourable, and distances not great, horse haulage is economical, but horses should not be employed on steep or long roads. Horses are attached to the corves either by chains or by some form of shafts, the latter being the better except on level roads. The cost of feeding a pit horse varies between 9*s.* and 13*s.* per week, according to the selection and market price of the food. Small engines driven by

electricity or compressed air are now being introduced at some collieries to do the work formerly done by horses.

Self-acting Inclines.—Coal is usually conveyed from a higher to a lower level by means of self-acting inclines, or "jinneys." A wheel or drum is fixed at the top of the incline, round which a rope passes. To one end of this rope the full run is attached, and to the other end the empty run. As the full run descends the incline, it pulls the empty run up by means of the rope.

FIG. 160.—Self-acting incline.

Fig. 160 shows a common arrangement of roads on a self-acting incline. There are pass-byes at top, bottom, and centre. The lower length is laid with two rails, and the upper with three.

A common form of jinney wheel is shown in Fig. 161. In order to prevent the rope from slipping when the brake is applied, two wheels are employed, the rope being passed round both; the upper and smaller of the two is provided with one groove, and the lower with two. The lower wheel is fitted with a powerful brake. The wheels and frame are set vertically between the two roads at the top of the incline. A self-acting incline should be steep at the top and flat at the bottom, to enable the load to start quickly and be easily stopped.

The motive power of a jinney consists of the gravity of the full run; the resistance is made up of the gravity of the empty run and of the rope, plus the friction of rope and full and empty runs. Thus the power of a jinney may be increased by putting additional corves on to the runs, and the resistance may be decreased by shortening its length and so reducing both friction and gravity of the rope.

Very long roads are usually divided into a series of jinneys, in order to proportionately reduce the weight of the rope.

The least inclination at which a jinney will work under given conditions can be calculated as follows:—

Find least inclination at which a jinney 800 yards long will

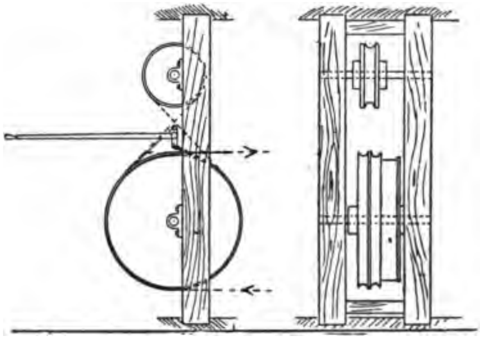


FIG. 161.—Jinney wheels and frame.

work, when the corves weigh 4 cwts. and carry 10 cwts., 15 corves to constitute a set, and rope to be $2\frac{1}{2}$ inches in circumference.

		lbs.
Weight of rope $\frac{C^2}{2} \times 800$	=	2500
,, empty set $15 \times 4 \times 112$	=	6720
,, full set $15 \times 14 \times 112$	=	23520
Friction of rope $(\frac{1}{28})\ \frac{2500}{28}$	=	89
,, full and empty sets $(\frac{1}{56})\ \frac{30240}{56}$	=	540

The force of gravity of the full corves must overcome the gravity of the empty set plus the frictional resistance of rope, full and empty sets. So that

$$\frac{23520}{g} = \frac{9220}{g} + 629 \text{ (friction resistances)}$$

$$\therefore\ 629 = \frac{23520}{g} - \frac{9220}{g}$$

$$\therefore\ 629 = \frac{14300}{g}$$

$$g = \frac{14300}{629} = 22 \cdot 7$$

So that the least gradient at which the jinney would work under the conditions given, is **1 in 22·7**.

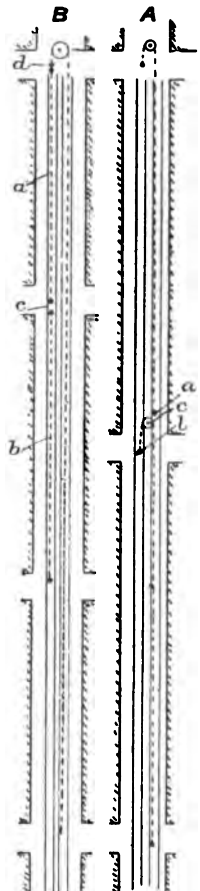


FIG. 162. — Arrangements for jinneying from several levels.

Jinneying from Two or More Levels.—It often happens that a self-acting incline has several intermediate landings to and from which corves have to be conveyed. Fig. 162 illustrates the two most common methods of arranging this. A (Fig. 156) shows the rope or chain divided into two sections. A small sheave is fixed at each landing in such a manner that it can be moved clear of the roads when not required. When serving the lower landing, the top portion of the rope is uncoupled, and lies idle, whilst the lower portion is in use. When the upper landing has to be served, the rope or chain is coupled at *a* and *l* by means of a D link, and the sheave *c* is moved out of the way.

In the method shown at B, one chain extends the whole length of the incline, and short lengths *a* and *b* extend from level to level, and can be uncoupled at *c* and *d*. The figure shows ropes in position to serve the second landing, the length *b* being uncoupled and lying idle.

Balance Inclines.—These are sometimes employed for conveying coal from a higher to a lower level. They are used chiefly in very steep seams on inclines having several landings, and also to assist trammers on steep roads. The road is laid with two pairs of rails, placed either side by side or one inside the other. The balance carriage travels on the one pair and the corves on the other. A rope passes round a pulley at the top

of the incline, and has a balance carriage attached to one of its ends ; this balance carriage is heavier than the empty and lighter than the full corf. When an empty corf has to be raised to any of the levels which communicate with the incline, the weight of the balance carriage, as it descends the hill, pulls up the empty corf which is attached to the other end of the rope. The empty corf is then changed for a full one whose weight is sufficient to pull the balance weight back again to the top of the incline.

Single-rope Haulage.—This method of haulage is only applicable when the dip of the road is sufficiently steep to enable the empty corves to run to the bottom of the hill and drag the rope after them by gravity.

The hauling engines may be placed either on the surface or underground. They should have a drum provided with an efficient brake, and capable of being thrown out of gear with the engine. To lower the empty corves, the engine is stationary and the drum thrown out of gear ; the corves run down the plane by gravity and take the rope with them, the speed being controlled by the brake. When they reach the bottom, the rope is changed from the empty to the full run, the drum thrown in gear, and the engines started. The engines are similar to winding engines, except that the drum is driven through gearing in order to reduce the speed of the rope to about 10 miles per hour.

The rope is carried on light steel rollers, about 6 inches in diameter.

The resistance the engine has to overcome is made up of—

$$\left.\begin{array}{r}\text{Gravity of coal and corves}\\ \text{gravity of rope}\end{array}\right\} = \frac{\text{total wt.} \times \text{vertical ht. of incline}}{\text{length of incline}}$$

$$\text{Friction of coal and corves} = \frac{\text{weight of coal and corves}}{56}.$$

$$\text{Friction of rope} = \frac{\text{weight of rope}}{28}$$

Example.—The total rise in a road 900 yards long is 150

feet: find the horse-power necessary to haul 10 tons of coal up it in $4\frac{1}{4}$ minutes, the hauling rope being $2\frac{1}{2}$ inches in circumference.

The weight of the corves is usually about one-half the weight of coal carried, so the total weight of coal and corves will be 15 tons = 33,600 lbs.

The weight per yard of a rope $2\frac{1}{2}$ inches in circumference is $\frac{2{\cdot}5 \times 2{\cdot}5}{2} = 3{\cdot}125$ lbs. (see p. 310)

The total length of rope is 900 yards, but the average length is $\frac{\text{maximum length} + \text{minimum length}}{2}$ = 450 yards, and the average weight is $450 \times 3{\cdot}125$ = 1,406

Average moving weight = 35,006

The length of the plane is 900 yards, and the height 50 yards, so that the resistance due to gravity is $\frac{50}{900} = \frac{1}{18}$ of total weight.

$$\text{Gravity of coal, corves, and rope} = \frac{35006}{18} = 1945 \text{ lbs.}$$
$$\text{Friction of corves} = \frac{33600}{56} = 600 \text{ ,,}$$
$$\text{Friction of rope} = \frac{1406}{28} = 50 \text{ ,,}$$
$$\text{Total resistance} = 2595 \text{ ,,}$$

The speed at which the load moves is $\frac{900 \times 3}{4\frac{1}{4}} = 635$ feet per minute, so that the

$$\text{Horse-power is } \frac{2595 \times 635}{33000} = 49{\cdot}9$$

The maximum horse-power is rather more than this, and the minimum rather less, on account of the varying weight of the rope. Taking the useful effect of the engines at 50 per cent., the necessary indicated horse-power is about 100.

This method of haulage is sometimes modified by having two drums and hauling one train of corves up whilst another runs down.

Drags.—The full run must be provided with a drag to throw the corves off the road if the rope or couplings break. This is a strong iron bar, pointed at its extremity, and dragged behind the last corf in each train. It may be attached to the axle, draw-bar, or to the body of the corf; if the trains runs back down the plane, the drag is driven into the floor, and should either stop the corves or throw them off the road, and so prevent their dashing down the plane at an ever-increasing velocity.

Gradient.—The least gradient upon which this method of haulage can be employed depends upon the coefficient of friction of the corves and upon the length of rope they have to pull behind them; in practice, a gradient of about 1 in 20 is the flattest that can be successfully dealt with. If the road is too flat, much loss of time and trouble is occasioned by the empty corves refusing to run in.

Main and Tail Rope Haulage.—When the road is undulating, or is not sufficiently steep to enable the empty corves to run in and take the rope with them, two ropes may be employed; one, known as the main rope, pulls the full corves to the pit-bottom; and the other, known as the tail rope, pulls the empty corves into the workings. The sets travel at a speed of ten or twelve miles per hour.

The general arrangements of a system of main and tail rope haulage are shown in Fig. 163. The engine is provided with two drums, *a* and *b*; *a* is for the main, and *b* for

FIG. 163.—Main and tail rope haulage.

the tail rope. Either of these drums can be put in gear with the engine, or can run free, and each is provided with a brake.

The tail rope is twice the length of the engine plane. It passes from the drum to the end of the plane, where it is taken round a return wheel and back to the pit-bottom. The return wheel may be either horizontal or vertical. In the former case it is set under the rails, and in the latter either at the road-side or between the roads in the pass-bye. The tail rope is carried from drum to return wheel on sheaves; these may be carried on roof, floor, or side.

In the figure the empty corves are being hauled to the far end of the engine plane. To do this the main drum *a* is thrown out of gear, and runs loose; the tail rope-drum *b* is in gear, and as the rope is wound upon it by the engine the corves are hauled along. The main rope is attached to the end corf in the train, so that it is dragged to the end of the plane, behind the run.

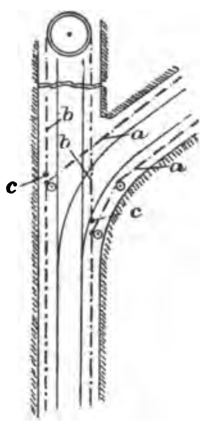


FIG. 164.—Main and tail rope haulage—branch.

When the pass-bye is reached, both ropes are changed on to the full train; the main-rope drum is thrown in gear, and the tail-rope drum allowed to run loose; so that when the engines are started the load is hauled to the pit-bottom by the main rope, and the tail rope dragged behind it in readiness for the next train of empties.

Branches.—There are several methods by which branches can be worked. Fig. 164 shows one method which is frequently adopted. The branch road is provided with a separate tail rope double its length, and working round a return wheel in the usual manner. The principal tail rope has in it two shackles, *c*, *c*, by means of which that part of the rope marked *b b* can be disconnected; these shackles are made very small and neat, to enable them to run round the return wheel.

The ends of the branch tail rope *a*, *a* are also fitted with

shackles. When the train is in the pit bottom, the shackles in the principal tail rope come just opposite those in the ends of the branch tail rope. If the train has to run into the branch, the shackles *c*, *c* are disconnected and connected to the shackles at the ends of the branch rope *a*, so that when the engine starts the rope *a* moves, whilst *b* lies idle.

The main and tail rope method of haulage can be applied to roads having varying gradients and many bends; it also possesses the great advantage of only requiring a single road, and for roads of great length there is no method by which a moderate quantity can be hauled so cheaply.

A very large output cannot be dealt with from a single road by the main and tail rope method, and chiefly for this reason it is seldom put down at new collieries, at any rate in the Midlands.

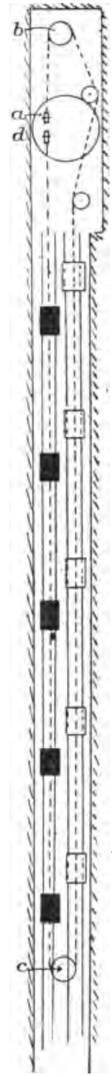


FIG. 165.—Endless-rope haulage.

Haulage by Endless Rope.—In this method of haulage a double road is required, and an endless rope extends from one end of the road to the other, and travels slowly along it. The general arrangement is shown in Fig. 165. The rope is taken down the shaft from engines on the surface, under the pulley *a* to the tightening wheel *b*, where the slack rope is taken up; from *b* it is led down the engine plane, round the return wheel *c*, and back to the pit bottom, where it passes under the pulley *d* and up the shaft to the engine. The corves are hung on to the rope either singly or in sets, and the rope may travel either above or below them.

Engines.—The engines may be placed

either on the surface or underground. If the shaft is of moderate depth, it is usual to fix the engines on the surface, and drive the haulage by means of a belt rope; but if the shaft is very deep, the haulage is best driven either by electric motors or by engines driven by compressed air, placed underground.

The gearing of the engines is required to give the rope a speed of from 2 to $3\frac{1}{2}$ miles per hour. For example, if an engine makes 70 revolutions per minute, and the rope wheel is $7\frac{1}{2}$ feet in diameter, what must the ratio of gearing be to give the rope a speed of $2\frac{1}{2}$ miles per hour?

$$2\tfrac{1}{2} \text{ miles per hour} = \frac{1760 \times 3 \times 2\frac{1}{2}}{60} = 220 \text{ feet per min.}$$

$$\text{Circumference of rope wheel } 7\tfrac{1}{2} \times 3{\cdot}141 = 23{\cdot}56 \text{ feet}$$

$$\text{Revs. of rope wheel per min.} = \frac{220}{23{\cdot}56} = 9{\cdot}34$$

$$\text{Ratio of gearing} = \frac{70}{9{\cdot}34} = 7{\cdot}49 \text{ to } 1$$

As the engines run in one direction only and at a uniform velocity, neither brake nor reversing gear is necessary, but a set of good governors should be provided.

Rope Wheel.—Many patent clip pulleys have been designed, but an ordinary taper pulley around which the rope makes four or five turns is generally adopted. An example of this class of pulley is shown in Fig. 166, which is a section through the rim. The pulley is made of cast iron, but is fitted with a steel liner, *a*, which can be taken out and renewed when worn. The rope goes on to the pulley at the larger and comes off at the smaller diameter, so that each coil has to slip down the pulley, and to enable this to be done smoothly the pulley is tapered.

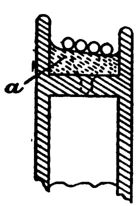


FIG. 166.—Rope wheel.

The steel liners are made of many different shapes and tapers; the one in the figure has a straight taper of 1 in 12.

Tightening Pulleys.—Arrangements should be made for taking up the slack in each rope. One form of tightening gear is shown in Fig. 167. The horizontal pulley *a* is mounted on the carriage *b*, which runs on a short length of rails. The rope passes round the pulley and strain is put upon it by means of the heavy weight *c*. Screws are sometimes employed instead of weights, but the latter are preferable, as they adapt themselves automatically to the varying stresses which are put upon the ropes. The tension carriages are sometimes placed at the far end of the haulage road, but more often close to the driving pulley.

Junctions.—The haulage on each road should be worked by an independent rope, so that one road may stand whilst the

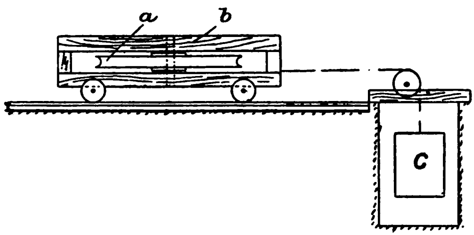


FIG. 167.—Endless-rope haulage—tightening pulley.

others work, and any accident only affects the road upon which it occurs.

The arrangements at an important junction are shown in Fig. 168. *a* is a vertical shaft, having keyed on to it the rope wheel *b*. Loose on the same shaft are the two rope pulleys *c* and *d*; each of these latter is provided with a friction clutch, by means of which it can be put into gear with the shaft *a*. The shaft continually revolves, as it is driven by the engine through the band rope, which passes four times round the fixed pulley *b*. The band rope does no actual hauling, but only serves to transmit the power from the engine to the shaft, and may be dispensed with if power is transmitted by electricity or compressed air.

The pulleys *c* and *d* each serve a different road, and can

be started or stopped at will by means of friction clutches. The arrangement shown in the figure is suitable for fixing near a pit bottom which has two main roads; a similar arrangement is placed at each branch road.

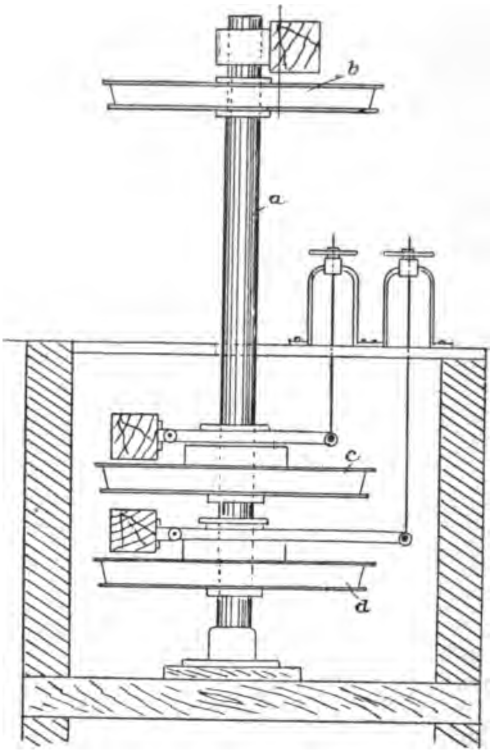


FIG. 168.—Endless-rope haulage—arrangements at junction.

Friction Clutches.—In Fig. 168, the shaft *a* continually revolves, but the pulleys *c* and *d* must be arranged either to revolve with the shaft or to run loose upon it. This is done at will by means of friction clutches. The principle upon which friction clutches act will be understood from Fig. 169. *a* is

the shaft upon which the pulley *b* runs loose; *c* is a cast-iron boss, which is keyed to the shaft *a*, and therefore revolves with it.

The pulley *b* is fitted with a flange, *d*, placed inside which is the casing *e*. This casing is connected to the boss *c*, and therefore revolves with the shaft. If the casing fits the flange loosely, it will revolve inside it, but by moving the sleeve *f* inwards, the casing is forced against the flange by the connecting rods *g*; the flange is thus held tightly to the casing and is pulled round with it. The sleeve is actuated by means of a hand-wheel and levers, and the pulley can be gradually thrown in or out of gear without stopping the shaft.

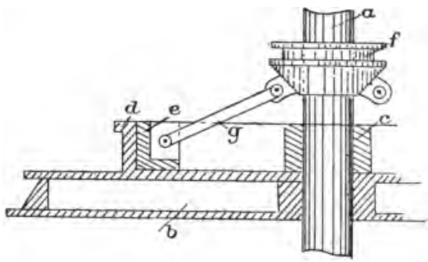


FIG. 169.—Friction clutch.

Attachment of Corves to Rope.—The corves may be attached to the rope either singly or in sets. When the rope is carried above the corves, the attachment is frequently made by means of a lashing chain about 10 feet long, made of iron about $\frac{3}{8}$ inch in diameter. Each end of the lashing chain carries a hook, one of which is hung on to the corf; the other end of the chain is twisted three or four times round the rope and brought through the hook. Clips are also used for overhead haulage, but chains have the advantage of requiring fewer binding pulleys, as the ropes need not be kept exactly in the centre of the road.

When the rope is carried under the corves, some form of clip must be used. For light gradients Fisher's clip, which is shown in Fig. 170, gives excellent results. The hook *a* is hung to the corf, and the hauling rope is passed through the jaws *b*.

To detach a corf, the box *c* is knocked up, and the jaws open at the hinge *d* and liberate the rope.

In a few cases the rope is carried at the side of the corf; this necessitates wider roads, and has no special advantages.

Curves.—Great care is required in taking endless-rope haulage round curves, especially where the gradients are heavy. The strain upon the rope tends to pull it into the side, and it has to be kept in its position in the centre of the road by

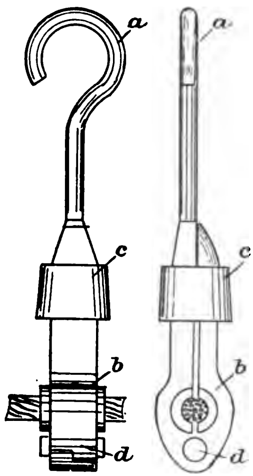


FIG. 170.—Fisher's clip.

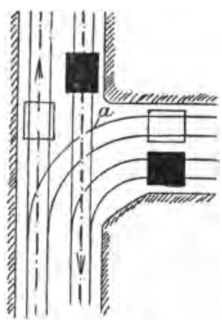


FIG. 171.—Endless-rope haulage —branch.

means of pulleys. A number of small vertical pulleys are usually employed. They are set almost touching each other, in order to divide the strain and keep the ropes between the rails. The pulleys should be set on strong sleepers, long enough to stretch across the whole width of the road, and pinned tightly to the sides.

Branches.—The haulage roads are fed by branches, at each of which some of the empty corves are taken off the rope and full corves substituted. An ordinary arrangement of a branch road is shown in Fig. 171. It will be noticed that the empty

corves have to pass across the road along which the full corves travel. If the ropes are carried overhead, they are raised at the branches to allow the corves to pass beneath them; but if the ropes are under the corves, the rails are cut at *a*, and the rope is carried a little below rail-level.

To avoid the trouble caused by taking the corves across one of the roads, subways are sometimes employed at busy branches. When this is done the full corves, instead of crossing one road, run underneath both, and are switched back when they reach the other side.

The size of engines necessary for an endless-rope haulage may be calculated as in the following example:—

Find the size of hauling engines necessary to haul 1000 tons in 9 hours up a road one mile long, having a total fall of 660 feet, taking the corves to weigh $4\frac{1}{2}$ cwts. and carry 10 cwts., the rope to be 3 inches in circumference, and the pressure of steam on the boilers to be 80 lbs. per square inch.

$$\text{Tons per hour, } \frac{1000}{9} = 111 \cdot 1$$

Taking the speed of the rope to be $2\frac{1}{2}$ miles per hour; as the road is one mile long, an amount of coal equal in weight to all the coal on the rope is landed at the top of incline $2\frac{1}{2}$ times an hour, hence the quantity on rope must average $\frac{111 \cdot 1}{2 \cdot 5}$ $= 44 \cdot 4$ tons $= 888$ cwts. Number of full corves on rope, $\frac{888}{10} = 88 \cdot 8$, say 89 full, and the same number empty.

Weight of rope per yard $\frac{3^2}{2} = 4 \cdot 5$ lbs.; total weight, $4 \cdot 5 \times 1760 \times 2 = 15{,}840$ lbs.

Total resistance = gravity of coal (the corves balance each other) plus friction of full and empty corves and friction of the whole length of rope.

Weight of coal in pounds, $44 \cdot 4 \times 2240 =$	99,456
Weight of corves in pounds, $89 \times 2 \times 4 \cdot 5 \times 112 =$	89,712
Total	189,168

The height of the plane is 660 feet, and the length is 5280 feet, so effect of gravity is $\frac{660}{5280} = \frac{1}{8}$ of load.

Resistance due to gravity of coal, $\frac{99456}{8}$	=	12,432
,, ,, friction of coal and corves, $\frac{189168}{56}$	=	3,378
,, ,, ,, rope, $\frac{15840}{28}$	=	566
Total resistance		16,376

The speed of rope is $2\frac{1}{2}$ miles per hour; that is, $2\frac{1}{2} \times 88 = 220$ feet per minute.

So that a resistance of 16,376 lbs. is being moved through a space of 220 feet per minute. Hence foot-lbs. of work done per minute are $16{,}376 \times 220$.

If the efficiency of the plant is 70 per cent., and a pair of engines are employed working at a piston speed of 360 feet per minute and having an average steam-pressure in the cylinder of half the pressure on the boiler, the diameter of their cylinders should be—

$$\sqrt{\frac{16376 \times 220 \times 100}{40 \times 360 \times 2 \times 0{\cdot}7854 \times 70}} = 15\tfrac{1}{8} \text{ inches}$$

Wherever mechanical haulage is employed, means of communication with the engine-man must be provided. This is now universally accomplished by the use of electric signals. The current is carried by bare wires fastened upon insulators, which are secured to props fixed alongside the road. By bringing these wires into contact with each other, or by connecting them by a metal conductor, a bell is rung in the engine-house. Signal wires can also be made to carry telephonic messages by connecting them to small portable telephones.

CHAPTER XXV.

PUMPING.

THE following figures relating to water should be committed to memory :—

1 cub. ft. of water weighs 62·5 lbs. and contains 6·25 galls.

1 cub. in. „ 0·0362 lb.

1 gallon „ 10 lbs. „ 277·27 cub. in.

1 foot head of water gives rise to a pressure of 0·434 lb. per square inch.

Feeders of Water.—Water may be met with either as temporary or as permanent feeders.

The primary source of all permanent feeders is rain. In England an average of about 2500 tons of water falls each year on every acre of land in the shape of rain. Part of this water is re-evaporated, part is carried out to sea by the streams and rivers, and part sinks into the ground. The proportion that sinks into the ground depends upon the porosity of the strata which form the surface of the district, and upon the contour of the country. If the rocks are very open in texture and are not interrupted by faults, the rain will penetrate them very rapidly, and if they are broken by coal workings or are pierced by shafts, the rain may find its way into a mine very soon after it falls. In some cases the effects of a heavy rainfall are felt in a few days, but in others large beds of porous rocks may act as reservoirs and so regulate the feeders that little, if any, variations are felt.

Temporary feeders occur when old workings having no external source of supply are tapped, or when pounds of water

contained in rocks which are entirely isolated by faults are met with.

The following pages describe the various methods of dealing with water when met with in mines.

The Syphon.—This is an appliance for raising water over an elevation and depositing it at a lower level. Fig. 172 shows the syphon as ordinarily applied. *b* is the suction end, and *c* the delivery; this latter must be at a lower level than the suction, and the vertical height over which the water is raised must be less than the length of the column of water which can be supported by atmospheric pressure. Under ordinary con-

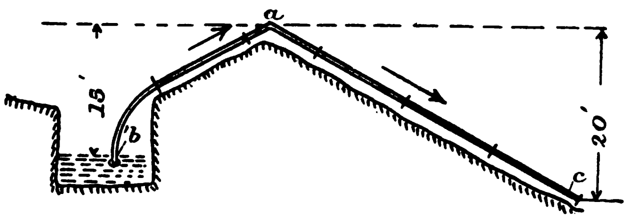


FIG. 172.—Syphon.

ditions the atmospheric pressure is about 15 lbs. per square inch (see Chapter XXI.).

The weight of a column of water 1 square inch in section and 1 foot high is 0·434 lb., hence the height of a column of water capable of being supported by the atmosphere is

$$\frac{15}{0·434} = 34·56 \text{ feet.}$$

Owing to the friction of the water as it flows through the pipes, and to the leakage of air at the joints, syphons will not work satisfactorily over an elevation of more than 20 to 25 feet.

The principle upon which a syphon works is as follows: If the pipe *bac* (Fig. 172) is closed by a valve at *a*, and each leg is full of water, the pressure of the atmosphere acts upwards on the open ends *b* and *c*, and is practically the same at each point, so that the upward pressure is 15 lbs. per square inch on *b* and on *c*. The downward pressure at *b* is

equal to that of a head of water 15 feet high, and is $15 \times 0{\cdot}434 = 6{\cdot}51$ lbs.; and the downward pressure at *c* is equal to that of a head of water 20 feet high, and is $20 \times 0{\cdot}434 = 8{\cdot}68$ lbs. per square inch. The net upward pressure on either end equals the total upward pressure minus the total downward pressure, which is $15 - 6{\cdot}51 = 8{\cdot}49$ lbs. per square inch on *b*, and $15 - 8{\cdot}68 = 6{\cdot}32$ lbs. per square inch on *c*. The effective pressure on the valve at *a* is the difference between these two pressures, and is $8{\cdot}49 - 6{\cdot}32 = 2{\cdot}17$ lbs. per square inch in the direction of the arrow; so that if the valve is opened the water flows from the higher to the lower pressure, that is, in the direction of the arrows.

To start a syphon, the whole of the pipes have to be filled with water. This may be done by fixing a hand-pump on to the delivery end, or by pouring in water through a funnel fixed on the pipes at their highest point. Great care must be taken to keep the pipe joints air-tight, or air will enter, as the pipes are under a partial vacuum. The pipes should have as uniform a gradient as possible, or air is apt to lodge in the high points and stop the flow by breaking the column of water.

Classification of Pumps.—Pumps used in mining may be divided into two classes: (1) Shaft pumps, and (2) dip pumps. Shaft pumps may be worked by rods actuated by engines placed on the surface, or they may be worked direct by steam, compressed air, or electricity. When worked direct, shaft pumps are similar in design to those used for pumping water from dip workings to the shaft bottom.

Pumps driven by rods take up much more room in a shaft than direct-driven pumps; they are, however, economical in steam consumption, and if the water should rise above them in the shaft, they are not affected, but under favourable conditions can be kept at work for a considerable time.

Bucket Pumps.—An ordinary bucket lift is shown in section in Fig. 173. *a* is the windbore, *b* the pump clack, *c* the clack door, *d* the bucket, *e* the knock-off joint, *f* the

pump rods, *g* the bucket door, *h* the working barrel, and *j* the pump trees.

Windbore.—The pump trees below the working barrel terminate in a windbore, through which the water which comes to the pump has to pass. The end of the pump is enlarged, closed at the bottom, and pierced with holes. All the water has to pass through these holes, which act as strainers and prevent solid substances from being drawn in.

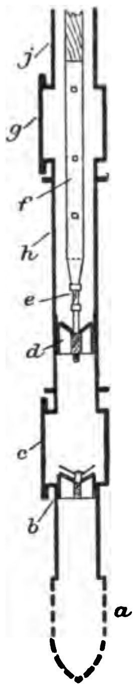


FIG. 173.—Bucket pump.

Clack.—There are many forms of clacks; the one shown below (*b*, Fig. 173) is suitable for pumps up to a diameter of about 15 inches. The clack has two lids opening upwards; these lids are formed of leather strengthened with iron plates. They open to allow the water to pass upwards to the working barrel, but close and prevent it from returning.

Clack Door.—The clacks are changed, when worn, through the clack door *c*. This is a strong cast-iron door bolted to the clack piece, a water-tight joint being made by means of an iron ring wrapped round with tarred flannel, or some other packing material.

Bucket.—This is usually similar in construction to the clack, and has lids opening upwards. Arrangements must be made to allow the bucket to move up and down in the working barrel without water passing it. The bucket shown in the figure is provided with a ring of leather, which projects above the bucket shell; the weight of the water upon this leather presses it tightly against the working barrel, and so prevents water passing it.

Knock-off Joint.—The bucket is connected to the rods by means of a wrought-iron knock-off joint. By knocking the two hoops up, the bucket is freed from the rods, and can be withdrawn through the bucket door when the pump is at the top of its stroke.

Rods or Spears.—Square pine rods are employed, the lengths being coupled by means of wrought-iron fish-plates and square through bolts. The rods for bucket lifts are wet; that is, they work inside the pipes, which are always full of water. The stress upon pump rods should not be more than about 5 cwts. per square inch sectional area.

Working Barrel.—This is a cast-iron pipe, turned inside to a true circle. If the water is corrosive, the working barrel should be lined with brass, as if the inside becomes rough the bucket leathers are destroyed very rapidly.

Pump Trees.—These are usually of cast iron, made in 9-feet lengths, and a little larger in diameter than the working barrel. By increasing the diameter of the pump trees, it is possible to draw the bucket through the pipes and change it at bank, in case the water rose above the bucket door.

Action of a Bucket Pump.—The action of a bucket pump is as follows: As the bucket is raised a vacuum is formed beneath it, and the atmospheric pressure forces water into the working barrel from the sump. The clack lids open for the upward passage of the water, but as soon as the upward stroke is completed they close and hold the water in the working barrel.

At the downstroke the water in the working barrel remains stationary, but passes through the lids in the bucket as it descends. At the next upstroke the water in the working barrel is raised the length of the stroke, because the bucket lids close, and if the pipes are full, a volume of water equal to the contents of the working barrel is discharged at the top, and at the same time water is drawn into the working barrel from the sump.

The vertical height from the level of the water in the sump to the top of the bucket, when at the highest point in its stroke, should not be more than about 25 feet, as the water is forced up by atmospheric pressure.

Bucket pumps will not satisfactorily raise water more than 100 yards in a single lift; if the shaft is deeper than this, two or more lifts should be employed, the bottom one delivering into a tank, from which the next takes its water.

The quantity of water delivered per minute by a bucket pump is equal in volume to the capacity of the working barrel (taking its length to be the length of the pump stroke) multiplied by the number of strokes per minute.

Example.—How many gallons per minute will an 18-inch bucket pump deliver, if the length of stroke is 7 feet 6 inches, and the number of strokes per minute $5\frac{1}{2}$?

Area of working barrel, $18^2 \times 0{\cdot}7854 = 254{\cdot}47$ sq. in.

Length of working barrel, 7 ft. 6 in. = 90 in.

Contents of working barrel, $90 \times 254{\cdot}47 = 22902{\cdot}3$ cub. in.

Quantity delivered in gallons per stroke, $\left.\dfrac{22902{\cdot}3}{277{\cdot}27}\right\} = 82{\cdot}5$

Gallons per minute, $82{\cdot}5 \times 5{\cdot}5 = 453{\cdot}75$

To get the actual quantity, a deduction of 10 per cent. should be made for leakage, and for the water that passes through the valves before they have time to close.

10 per cent. of $453{\cdot}75 = \underline{45{\cdot}37}$

Quantity pumped per minute = 408 gallons

Calculations as to the capacity of pumps may be worked out more quickly by the following formula:—

$$G = d^2 \times 0{\cdot}034$$

where G = gallons per foot of stroke,

d = diameter of bucket or ram in inches.

Working out the example given above by this formula, we get—

$$G = 18^2 \times 0{\cdot}034 = 324 \times 0{\cdot}034 = 11$$

feet per minute, $7{\cdot}5 \times 5{\cdot}5 = 41{\cdot}25$

gallons per minute, $41{\cdot}25 \times 11 = 453{\cdot}75$

deduct 10 per cent., $453{\cdot}75 - 45{\cdot}37 = 408$

Ram Pumps.—A single-acting ram of ordinary type is shown in Fig. 174. *a* is the suction pipe, *b* the bottom or

suction clack, *c* the top or delivery clack, *d* the ram or plunger, *e* the stuffing-box, *f* the pump rods, *g* the rising main, and *h* the barrel.

Clacks.—Both the clacks *b* and *c* are of the same design, and each is fitted with lids opening upwards. When the lift

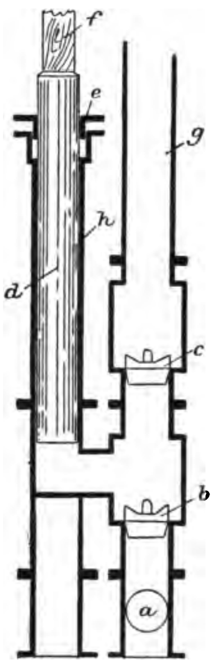


FIG. 174.—Ram pump.

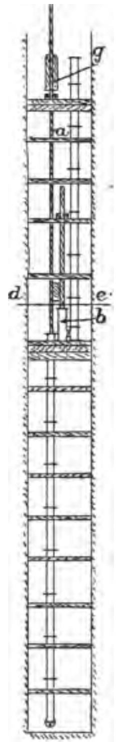


FIG. 175.—Pumping in two lifts.

is very long, double-beat or equilibrium valves are often employed; they give a large opening with a low lift, and close with little shock. It is now becoming common to employ a number of small valves instead of one large one.

Rams and Stuffing-boxes.—The ram *d* is a shell of cast iron, having its outer surface turned in a lathe. It is secured to the end of the pump rods *f*, and passes through the stuffing-box, which is packed with hemp or hydraulic packing, to prevent the leakage of water from the barrel. The stuffing-box and gland are usually lined with brass, and the ram may also be brass lined if the water is very corrosive.

Action of a Ram Pump.—As the ram makes its upstroke, water is forced through the bottom clack into the working barrel by atmospheric pressure. When the downstroke commences, the lids of the bottom clack close, and the ram forces the water in the barrel through the top clack into the rising main.

Pumping by Several Lifts.—Rams similar to the above are now made to force water to a considerable height in one lift; formerly it was the practice to limit the height of each lift to about 100 yards, and to pump from a deep shaft by a series of lifts. This is convenient where feeders of water are met with at different points in the shaft, as they can be dealt with by increasing the size of the rams.

A bucket pump does practically the whole of its work at the upstroke, whereas a ram delivers water at the downstroke only, so that by combining the two, the work done at each stroke can be to some extent equalized.

A common arrangement is to have a bucket lift at the bottom of the shaft, and ram pumps above.

The arrangement of rods and pumps in a shaft where the pumping is being done in two lifts, one a bucket and the other a ram, is shown in elevation in Fig. 175 and in plan in Fig. 176, which is an enlarged section on line *de*, Fig. 175. The main rod *a* is taken down the shaft to the bucket pump, whilst the ram *b* is driven by an offset rod.

Rods and pipes are secured by buntons placed across the shaft and let into the sides.

Catch-pieces, *g*, are bolted on to the rods at intervals, in order to catch them in case the pumps missed a stroke, or the

rods broke. Strong timbers or girders are let into the shaft just below the bottom of the catch-pieces, to hold the rods if they should fall.

When rams are worked by single-acting Cornish engines, the engines raise the rods and ram, but the downstroke—that is, the working stroke—is accomplished entirely by the weight of the rods. In very deep shafts the rods may have a considerable excess of weight, and this must be counterbalanced. Fig. 177 shows a balance bob, which may be placed either on the surface or in a chamber got out for it in the shaft side. One end of the tee bob is coupled to the pump spears by

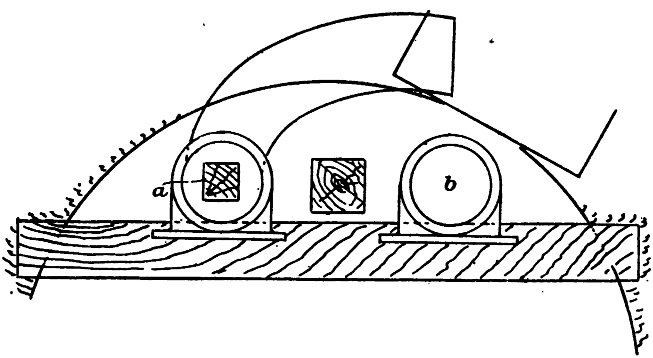


FIG. 176.—Section showing ram and bucket lifts.

a timber connecting rod, and the weight is secured to the other end.

The weight required to force a ram down may be calculated as follows:—

A ram 22 inches in diameter is delivering water against a head of 500 feet: find total weight of rods necessary.

Area of ram, $22^2 \times 0{\cdot}7854 = 380{\cdot}13$ sq. in.

Pressure per square inch on ram, $500 \times 0{\cdot}434 = 217$ lbs.

Total pressure on ram, $217 \times 380{\cdot}13 = 82{,}488$ lbs.

Add 10 per cent. for friction in stuffing-box, etc. $= 8{,}249$ lbs.

Total pressure required $= 90{,}737$ lbs.

This amounts to 40 tons 10 cwts., which is equal to over $1\frac{1}{2}$ cwt. per lineal foot of the rods.

Pump rods for deep shafts should be made largest at the shaft top, where the weight upon them is the greatest.

Arrangement of Engines to work Shaft Pumps.—Shaft pumps may be actuated by either rotary or reciprocating engines. The Cornish engine is a good example of the re-

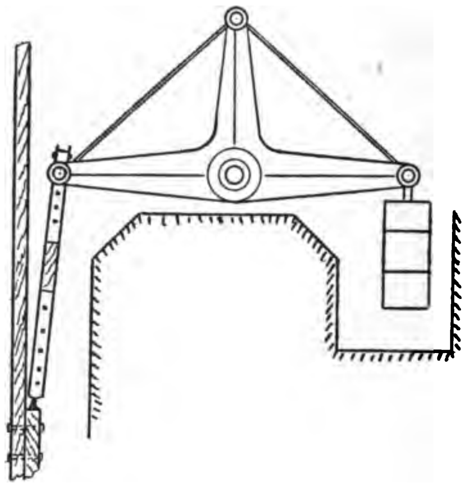


FIG. 177.—Balance bob.

ciprocating type. The pump rods are hung at one end of a beam, and the piston rod is coupled to the other; the engines make a pause at the end of each stroke, giving the pump valves time to close, and ensuring steady working.

Fig. 178 shows a reciprocating horizontal engine arranged to work two buckets or two rams by separate rods; both pumps may be at the pit bottom and deliver to the surface, or one may be at the pit bottom and the other halfway up, pumping the water in two lifts. The pump rods are hung from the horizontal limbs of the two bell cranks, and the

engine piston rod is coupled to the vertical legs; by this arrangement the strain on the engines is balanced, as one rod is making a downstroke while the other makes an upstroke.

A balanced arrangement suitable for a rotary engine is

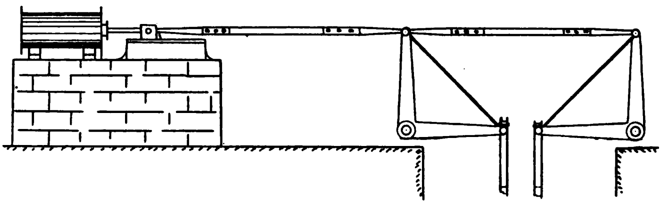


FIG. 178.—Horizontal engine working two lifts.

shown in Fig. 179. Here the two bell cranks *a* and *b* are placed side by side, and driven by cranks from either side of the large spur-wheel *c*, *c* being driven through gearing by a horizontal engine.

Capacity of Single-acting Pumps.—The pumps shown in Figs. 173 and 174 are single-acting—that is, they do not deliver water at both up and down strokes. The effective speed

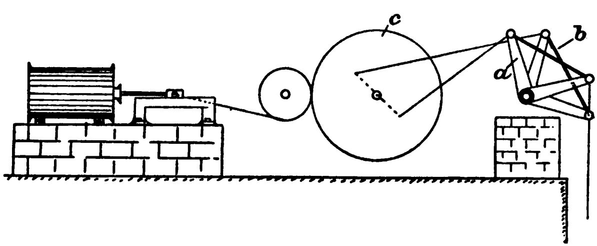


FIG. 179.—Pumps driven by rotary engine.

at which pumps of this class run is about 50 feet per minute; the ram or bucket actually travels twice this distance, but delivers water during only one-half of its time.

The size of a single-acting pump required to deliver a given quantity of water may be found as follows:—

Find diameter of bucket pump necessary to pump 600 gallons per minute.

Effective speed of ram = 50 feet per minute

Gallons per foot of stroke, $\frac{660}{50} = 13{\cdot}2$.

By formula given on p. 350, $G = d^2 \times 0{\cdot}034$

Hence $d = \sqrt{\frac{G}{0{\cdot}034}}$, and $\sqrt{\frac{13{\cdot}2}{0{\cdot}034}} = 19{\cdot}7$ inches diameter

Direct-acting Pumps.—A section through a double-acting pump is shown in Fig. 180. *a* is the pump rod, which is an

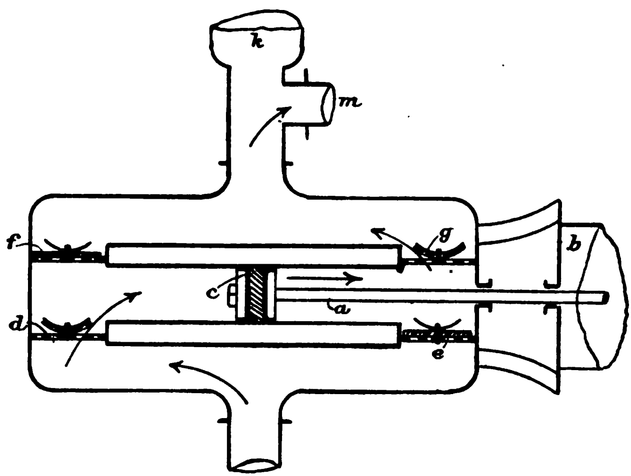


FIG. 180.—Double-acting piston pump.

extension of the engine piston rod; *b* is the steam cylinder, which is fixed on the same bed-plate as the pump; *c* is the pump bucket; *d* and *e* are the suction, and *f* and *g* the delivery valves; *k* is the air-vessel; and *m* the delivery branch, on to which is bolted the rising main.

Bucket.—This is really a piston, and differs from the bucket used in single-acting pumps by having no valves. It consists

of a solid cast-iron block fitted with a cup leather at either side; these cup leathers are forced against the pump barrel by the water-pressure, and so prevent the water from passing the bucket.

Valves.—The valves shown in sketch consist of flat rubber discs working upon grids, and provided with a guard to prevent them from opening too far. The disc is fixed at the centre, and is raised by the pressure of the water upon its underside, whilst pressure from above closes the valves by forcing the rubber down on the grid.

Valves of this type are very efficient for lifts of moderate height. When the head of water is great, groups of small circular brass valves are employed.

Air-vessel.—As water is incompressible, the flow in the rising main would stop and start at each stroke of the pump were it not for the air-vessel. This would cause a loss of power, owing to the inertia of the water, and would lead to an irregular discharge, and to greatly varying strains on the pump.

The air-vessel consists of a cast-iron vessel, closed at the top and fixed vertically between pump and rising main. It acts as a regulator by interposing an elastic body of air between the pump and the rising main.

As the pump delivers its water, the air in the upper part of the air-vessel (*k*, Fig. 180) is compressed, and part of the water is driven into the air-vessel. The moment the pump stops to reverse its stroke the pressure is relieved, and the air in the air-vessel expands and drives part of the water out of the vessel into the rising main. In this way the flow of water is made fairly constant, and as the water is always kept in motion, the pump has not to start it from a state of rest at each stroke.

Air-vessels on large pumps are sometimes charged with air by means of a small air-pump, but those on smaller pumps are charged automatically, the air which is drawn into the pump with the water rising to the highest point.

Pumps similar to that shown in Fig. 180 are not suitable

for high lifts or for bad water. When the leathers become worn the water slips past the bucket, and as this leakage is internal, it cannot be seen, and is difficult to detect. When the lift is high and the pump barrel roughened by wear or corrosive water, the leathers may require changing every few hours.

Action.—Steam is admitted into the cylinder *b* (Fig. 180) by valves, usually worked by some form of tappit. The steam presses first on one side of the piston, forcing it to one end of the cylinder and then to the other side, driving it back, and as the pump bucket is coupled to the steam-piston, it moves to and fro with it.

When the pump bucket is moving in the direction of the arrow, water flows through the suction valve *d*, and fills the space behind the bucket, whilst the water in front of the bucket is forced by it through the delivery valve *g*. In the return stroke, water is drawn in through the valve *e*, and expelled into the rising main through the delivery valve *f*.

Pumps of this class run at a piston speed of about 100 feet per minute; the size of pump and steam-cylinder for a given duty can be calculated as follows :—

Example.—Find size of pump and engine to deliver 300 gallons per minute against a head of 150 feet, the available steam-pressure being 60 lbs.

Gallons per minute	300
Add 10 per cent. for slip	30
	330

Gallons per foot of stroke, $\frac{330}{100} = 3\cdot3$

Diameter of pump, $\sqrt{\frac{3\cdot3}{0\cdot034}} = 9\cdot85$, say 10 in.

Pressure per sq. inch on bucket, $150 \times 0\cdot434 = 65\cdot1$ lbs.

Area of bucket, $10^2 \times 0\cdot7854 = 78\cdot54$ in.

Total pressure on bucket, $78\cdot54 \times 65\cdot1 = 5113$ lbs.

Add $\frac{1}{3}$ for friction in pump and in pipes, $\frac{5113}{3} = 1704$ lbs.

Total resistance $= 6817$ lbs.

$$\left.\begin{array}{l}\text{Take average pressure on piston at } \tfrac{2}{3} \\ \quad \text{boiler pressure, } \dfrac{60 \times 2}{40}\end{array}\right\} = 40 \text{ lbs.}$$

$$\text{Area of steam-cylinder, } \frac{6817}{40} = 170{\cdot}4 \text{ sq. in.}$$

$$\text{Diameter of steam-cylinder } \sqrt{\frac{170{\cdot}4}{0{\cdot}7854}} = 14\tfrac{3}{4}, \text{ say } 15 \text{ in.}$$

This shows that the pump bucket should be 10 inches in diameter and the steam-cylinder 15 inches, the piston speed being 100 feet per minute.

Pipes.—The pipes should be of ample size, because the friction of water when passing through pipes varies with the square of the velocity. They should be of such a diameter as to keep the velocity of the water down to from 200 to 250 feet per minute.

Example.—Find diameter of pipes required for pump given in last example, allowing a velocity of 220 feet per minute.

$$\text{Cubic feet of water per minute, } \frac{300}{6{\cdot}25} = 48$$

$$\text{Area of pipe} = \frac{\text{quantity}}{\text{velocity}} = \tfrac{48}{220} = 0{\cdot}2182 \text{ sq. ft.}$$

$$\text{Area of pipe in square inches, } 144 \times 0{\cdot}2182 = 31{\cdot}42 \text{ sq. in.}$$

$$\text{Diameter of pipe} = \sqrt{\frac{31{\cdot}42}{0{\cdot}7854}} = 6\tfrac{3}{8} \text{ in.}$$

When a pipe line is inclined, the pressure is the same as if the pipes were vertical, and of a length equal to the total rise of the incline; but as the length is greater for a given head when inclined, the friction is correspondingly greater, and allowance should be made for this by slightly increasing the area of the pipes.

Double-acting Ram Pumps.—In the arrangement shown in Fig. 181, two solid plungers, *a* and *b*, are connected to each other, and are given a reciprocating motion by means of a forked connecting rod, which is driven by a crank from a spur-wheel. The action of the pump is similar to that of

a bucket or piston pump, one ram delivering whilst the other is drawing in water through the suction. The two rams work

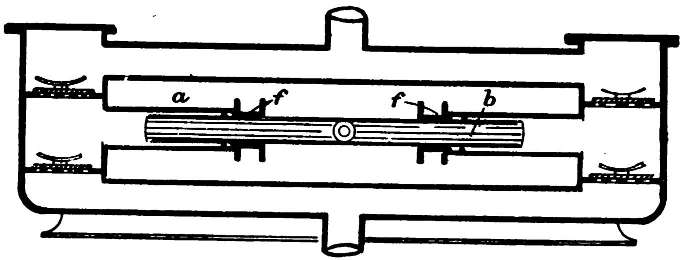


FIG. 181.—Double-acting ram pump.

through the stuffing-boxes *f*, *f*, and as they are externally packed, leakage is readily detected. Ram pumps are less

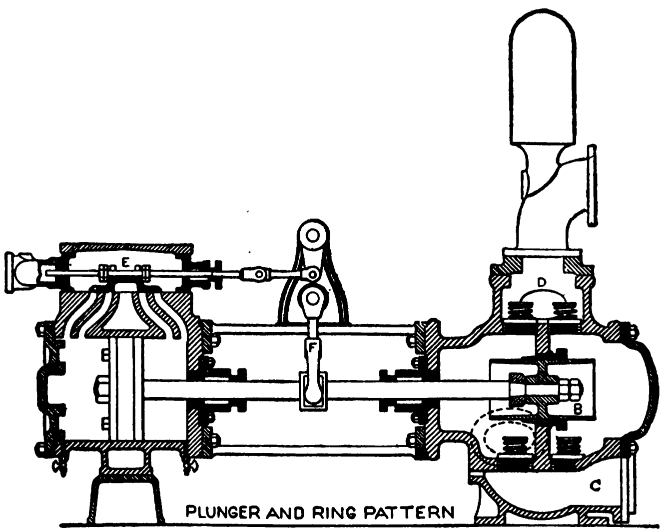


FIG. 182.—Section through one side of a Worthington pump.

compact and more expensive than pumps of the bucket or piston type, but are much better suited for high lifts.

Duplex Pumps.—This arrangement consists of two pumps placed side by side and mounted upon one bed-plate. The valves of one of the pumps are operated by levers driven by the other; thus one piston gives steam to the other, then finishes its own stroke and pauses until its valve is opened in its turn by the other engine.

Fig. 182 is a sectional view of one side of a Worthington pump. E is the slide valve, driven by a lever from the engine by its side; F is the lever which operates the valve of the other engine. The double-acting plunger B works through a deep metallic packing ring; the water enters the pump through the suction C, and has a nearly straight course into the delivery D.

Three-throw Pumps.—These pumps are suitable for being driven either by electricity or by wire ropes, and are now very largely employed. They consist of three ram pumps placed side by side on one bed, and driven by connecting rods from three cranks, which are arranged on one shaft and set at an angle of 120° with each other. By having three pumps, the strain on the driving shaft is kept almost uniform, and the delivery of the water nearly constant. Fig. 183 shows a three-throw Deane pump, as made by the Worthington Pump Company, suitable for being driven by a belt from an electric motor.

Hydraulic Pumps.—In general arrangements hydraulic pumps are somewhat similar to the pump shown in Fig. 180, but the motive power is water at a high pressure instead of steam or compressed air. The hydraulic cylinder, which corresponds to the steam-cylinder *b*, Fig. 180, is usually of smaller diameter than the pump, as the water-pressure which drives the pump is in most cases much greater than the pressure against which the pump delivers its water. The pressure of the water employed for motive power may be either obtained artificially by a force pump, or from a natural head.

When worked by a natural head, a small volume of water at a high pressure is employed to raise a larger volume against a lower pressure. If, for example, water were piped down

a shaft 400 yards deep, the pressure at the bottom of the pipe would be $400 \times 3 \times 0{\cdot}434 = 520{\cdot}8$ lbs. per square inch.

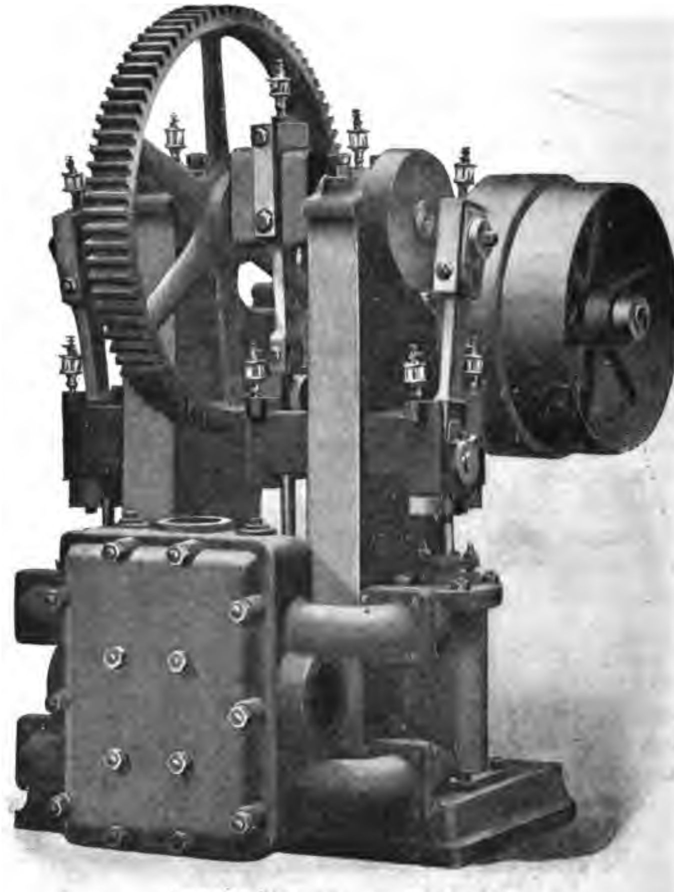


FIG. 183.—Double-acting vertical three-throw Deane pump.

If the volume of water amounted to 50 gallons per minute, the horse-power due to this weight of water falling this distance

would be $\frac{400 \times 3 \times 50 \times 10}{33000} = 18{\cdot}2$ (weight in pounds multiplied by vertical distance in feet divided by 33,000). About half of this power would be absorbed by frictional and other losses, but the remainder could be utilized to pump water from dip workings to the shaft bottom.

Pulsometers.—These pumps are designed to lift large volumes of water against a low pressure. They are seldom seen in mines, but are much used in coal washeries, and sometimes in sinking pits. They have no movable parts, with the exception of simple automatic valves, so are not apt to get out of order. They will pump very dirty water without trouble, but are rather extravagant in steam, and cannot deal with high lifts. Fig. 184 is a sectional view of a pulsometer. It consists of a large pear-shaped casting, divided longitudinally as shown. *a* is the suction pipe, *b* and *c* the inlet valves, *d* and *e* the outlet or delivery branches, *f* the steam-valve, and *g* the steam inlet.

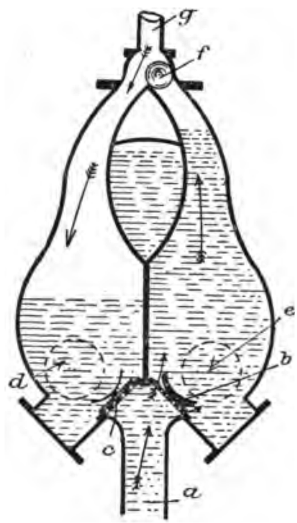


FIG. 184.—Pulsometer pump.

To start a pulsometer, it is filled with water, and the steam turned on. The steam-valve *f* is a rubber ball, and is so arranged that it must cover one or other of the openings, but cannot cover them both, so that as soon as the steam is turned on, it rushes through the opening which happens to be uncovered into one of the chambers, and by its superior pressure forces the water in the chamber through the delivery branch into the rising main. The two delivery branches unite, and are provided with valves, which open for the passage

of water from the pump to the rising main, but close to prevent its return. As the steam enters, it presses the water down gently and without agitation until its surface is below the top of the outlet *d*, but as soon as this point is reached, violent agitation is set up, causing the water to mingle with the steam in the chamber; this results in the steam being condensed, and a partial vacuum being formed. The effect of the vacuum is to pull the ball valve over, closing the chamber to the steam, and to draw in water through the suction pipe. When the valve is pulled over, steam enters the other chamber, and operates there in exactly the same way. Both inlet and outlet valves are of rubber, and may be arranged in such a manner that when they are opened, the full area of the valve opening is unobstructed.

Centrifugal Pumps.—A large volume of water may be raised to a moderate height by means of a centrifugal pump. These pumps are similar in principle to centrifugal fans. Water enters the casing at the centre, and is whirled round by vanes which rotate at a high velocity, and is expelled at the circumference by centrifugal force. The most convenient method of driving a centrifugal pump, when applied to drain dip workings, is by means of an electric motor, the shaft of the armature being coupled direct to the pump spindle.

Pumps for Sinking Shafts.—Sinking pits may be drained either by ordinary bucket lifts, similar to that shown in Fig. 173, hung on ground spears, and lowered to follow the shaft down, as explained in Chapter VII., or direct-acting steam-pumps of special design may be employed.

Direct-acting sinking pumps usually consist of a vertical arrangement of steam cylinder and rams, designed to occupy as little space as possible. The method of suspending the pumps and pipes has already been explained. Pulsometers have also been employed for draining sinking pits, several lifts being used, pumping to and from cisterns placed in the shaft. The chief difficulty met with when pumping water

from a sinking pit by a series of independent pumps is the regulation of the water. If one pump delivers more water than the pump above, part of the water must run back, and if the upper pump delivers more water than the lower, it will pump the cistern dry, and run on air.

Winding Water.—Small quantities of water may be dealt with by the winding engine. The water is allowed to accumulate in the sump and water-levels during the day, and is wound to the surface at night. An iron water-barrel is either hung below the cage, or is provided with wheels and run on to the cage in place of the corves, and is dipped into the sump. At the bottom of the water-barrel a valve opening inwards is fixed, through which the water enters when the cage is dipped. The discharge at the surface is effected by a lever, one end of which is connected to the valve by a rod; this lever strikes a bar when the cage is drawn above bank-level, and opens the valve.

Except when the quantity of water is small, winding should not be resorted to, as the shaft and fittings are damaged by the escaping water and by the vibration.

When a shaft is used exclusively for water winding, cigar-shaped barrels are often employed, as they run steadily, and enter the water without shock.

Riedler Pumps.—The chief feature of this class of pump is that the valves, instead of being left to close by the reversal of the stroke, are closed mechanically just at the proper moment. By this means these pumps can be run at a very high speed, without risk of injury. A piston speed of 300 to 400 feet per minute can be attained, which, of course, results in a large reduction in the size of pump required for a given duty.

CHAPTER XXVI.

SURFACE ARRANGEMENTS.

Engine-houses and Boilers.—The position of the engines, boilers, and other surface erections should be carefully chosen, so as to allow ample space, but with due regard to convenience and economy.

The boilers should all be set in a range, and room provided for extension; they should be placed with the view of avoiding long ranges of steam-pipes, and provision should be made for the economical conveyance of coal to the fire-holes, and of ashes from the ashpits. Each winding engine should have a separate house, but whenever possible all the other engines should be under one roof. It is now becoming the practice to drive haulage, screens, pumps, washeries, etc., by electric motors, so that the only engines absolutely necessary are the winding, electric generator, and fan engines, and shortly fans also will no doubt be driven by electric motors. Some of the older collieries have an enormous number of small engines running for various purposes, but this leads to large steam consumption, on account of the great length of steam-pipes and inefficient working of small engines.

Shops and Stores.—The workshops and store-rooms should be built in one block, and should have a waggon road running alongside them, and a corf road running into each shop, so that heavy machinery can be loaded on a corf in the shops, and sent straight down the pit.

The shops should consist of carpenters', smiths', and fitters' shops.

The blacksmiths' shop should contain single and double hearths, and should be provided with a good steam hammer; the fitting shop should contain lathes, shaping, drilling, and shearing machines.

A small foundry is a most valuable adjunct to a large

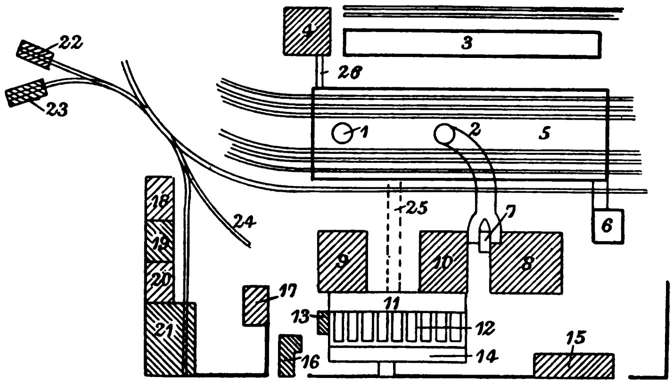


FIG. 185.—Surface arrangements.

1. Downcast shaft.
2. Upcast shaft.
3. Coke ovens.
4. Coal washer.
5. Screens.
6. Landsale shoot.
7. Fan.
8. Power house (dynamos, fan, engines, etc.).
9. Winding engines, downcast shaft.
10. Winding engines, upcast shaft.
11. Fireholes.
12. Boilers.
13. Feed pumps.
14. Economizers, in main flue.
15. Offices.
16. Time office and lamp-room.
17. Saw-mill.
18. Carpenters' shop.
19. Blacksmiths' shop.
20. Fitters' shop.
21. Stores.
22. Locomotive shed.
23. Waggon-repairing shed.
24. Timber yard.
25. Conveyor to fire-holes.
26. Conveyor to washer.

colliery, as repairs have often to be made as quickly as possible, and much time may be lost when small castings have to be made at a foundry situated some distance away from the colliery.

An example showing a compact arrangement of the surface erections necessary for a large colliery is given in Fig. 185.

Sidings.—Ample and well laid out sidings are of the utmost importance in dealing expeditiously with a large output. An example of the general arrangement of colliery sidings is shown in Fig. 186. These sidings are arranged to work entirely by gravity, so that the colliery company require no locomotive. The railway company's locomotive pushes the empty waggons into the empty sidings, which have a grade of about 1 in 60 towards the screens. When empty waggons are required, they are lowered by their brakes to the screens where they are loaded, and when full they are lowered into the full sidings, from whence they gravitate over the weighing machine, and are taken away by the railway company.

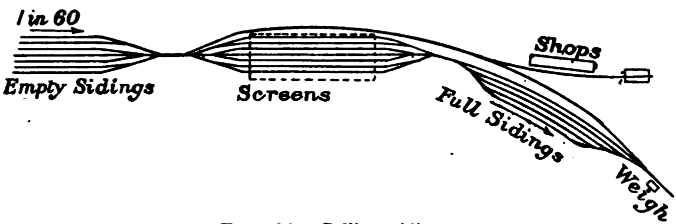


FIG. 186.—Colliery sidings.

Screens.—The pit bank should be raised from 20 to 30 feet above the level of the sidings, to enable the coal to gravitate over the screens and belts into the waggons. Modern pit banks are in almost every case constructed of iron or steel, ordinary steel girders of H section being usually employed, though some pit banks are built on cast-iron columns. It is desirable to have landings at the siding level as well as at the level of the pit bank; the former being employed for winding horses, timber, etc., and the latter for coal.

Banking the Corves.—The arrangements at a pit top must be well designed, in order to cope with the large outputs which are now required; there are collieries now at work which are turning over 500 corves per hour from one shaft. The full and empty corves should run on separate roads, and have a definite direction, all the roads being graded to enable

the corves to run without manual labour. A good form of pit bank is shown in Fig. 187, the direction in which the corves circulate being indicated by arrows. When the full corves are pushed off the cage, they run down to the switch-back *b* in Fig. 187; the road across the points rises sharply, and the corves mount the incline on account of the momentum they acquire in running down the grade. They run into spring buffers at the dead end, and are switched back automatically on to the road leading to the weighing machine. They are steadied at the machine by a man or lad, who pushes them off the platform when weighed; they then run to the tippler, and are tipped and pushed out at the other end, whence they run to the point A, which is the foot of the creeper. The point A is considerably lower than the

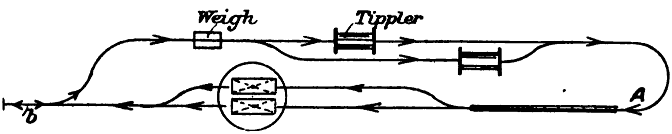


FIG. 187.—Arrangement of roads on pit bank.

cage bottom, because the corves have been running downhill for the whole distance; they are, therefore, raised by the creeper to a point above the level of the cage. From the creeper top they run into one or other of the roads from which the empty corves are run on to the cage; as a corf runs into one of the roads it pushes back a lever which moves the points to turn the next corf into the other road, so that a corf runs into each road alternately.

The weak point in the arrangement shown in the figure is that it provides no stand-room for corves, so that winding would have to be suspended whilst any small accident that might occur to the screens was being remedied.

Most cages have more than one deck, and the corves on each deck should be changed simultaneously. For this purpose platforms must be erected at the level of each deck, and arrangements made for conveying the corves to and from

the platforms. In the example given in Fig. 187, the empties might be raised to the upper platforms by additional creepers, and the full corves run to the weighing machine by separate roads. As these roads would be steep, the velocity of the corves would have to be checked; this is usually done by fixing long springs on either side of the road. When the corves pass between the springs they must push them apart, and are almost pulled up by the pressure.

Fowler's Hydraulic Decking Arrangement.—This is an arrangement for changing several decks simultaneously by

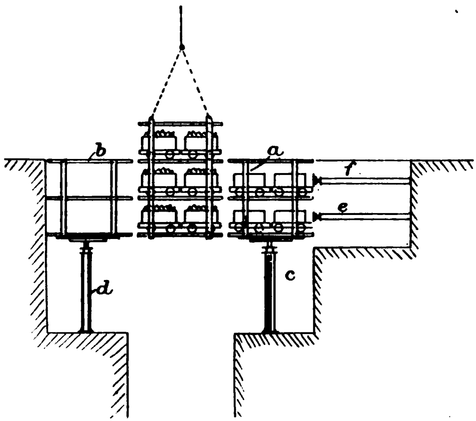


FIG. 188.—Fowler's hydraulic cage decking arrangement.

means of hydraulic rams. Fig. 188 shows the arrangement as adapted for a three-decked cage. The corves on the upper deck are changed by hand, and on the two lower decks by hydraulic rams. *a* and *b* are subsidiary cages resting on the rams *c* and *d*; *e* and *f* are the horizontal rams by which the cages are loaded. When the winding cage reaches the bank, the corves on the top deck are changed by hand, and whilst this is being done the empty corves, which have previously been placed on the cage *a*, are pushed off it on to the winding cage by the rams *e* and *f*, the full corves at the same time

being pushed off the winding cage on to the subsidiary cage *b*. The winding cage then descends, and whilst it is in the shaft the subsidiary cages are raised to bank, deck by deck, by means of the rams *c* and *d*, and fresh empty corves are pushed on to *a*, and the full ones removed from *b*. Both subsidiary cages are then lowered to the position shown in Fig. 188, in readiness for the next load. Four subsidiary cages are employed in all, two for each of the winding cages.

Creepers.—For raising corves from one level to another either hoists or creepers may be employed; creepers are now more common, as they are automatic in action, whereas the hoist requires a lad to drive it.

A creeper consists of an endless chain, which travels slowly up an incline, and runs in an iron trough. The chain is composed of flat links, some of which are provided with "fingers," or projections. The "fingers" stand above the top of the trough, whilst the chain itself is kept in the trough by flanges. When a corf runs to the bottom of the incline, one of its axles is caught by one of the fingers, and the corf is dragged up the incline, beyond the top of which the road is set with a slight gradient downward. The chain delivers the corf on to the top of the incline, and is then led below the rail level, so that the "finger" leaves the axle and the corf is free to run.

A hoist in its common form consists of a vertical cylinder carrying a light cage or platform on its piston rod, the load being raised by the admission of steam under the piston. Hoists take up less room than creepers, but require more steam and labour.

Tipplers.—The coal is emptied from the corves to the screen by means of tipplers. End tipplers are arranged to empty the coal over the end of the corf, and side tipplers over the side.

End Tipplers.—These consist of a framework or platform of iron suspended by pivots, and free to turn through the whole

or part of a circle. The corf is pushed on to rails on the tippler, and the tippler is revolved, discharging the contents of the corf over its end. The axis upon which the tippler turns is fixed parallel to the ends of the corf. End tipplers are now seldom used in modern screening plants, the objections being, that the corf has to be pushed in and pulled back out of the tippler. This results in loss of time and adds to the labour; moreover, the coal has to fall a considerable distance on to the screen, which leads to unnecessary breakage, especially when it is of a tender nature.

Side Tipplers.—An end elevation of a side tippler is shown in Fig. 189.

The tippler consists of two cast-iron rings, *a*, connected by

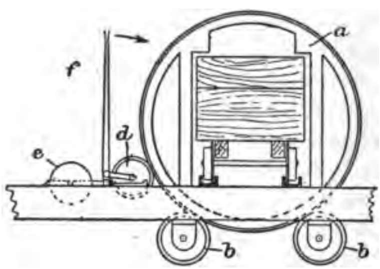


FIG. 189.—Side tippler.

ironwork, and provided with rails for the corves to run on and projections to hold them in position when being tipped. The rings *a* are carried by the rollers *b*, *b*, upon which they are free to revolve. The wheel *d* is driven by a belt from the shafting which drives the screens, and revolves continuously. *d* is carried on sliding pedestals, and its circumference is grooved to fit the edge of the ring *a*. When the tippler is out of action, the wheel *d* is kept clear of the ring *a* by means of the balance weight *e*. To start the tippler, the lever *f* is pushed in the direction of the arrow; this forces the sliding pedestal towards the tippler, and the grooved wheel *d* binds against the ring *a* and carries it round as it revolves. The tippler is

turned through a whole revolution, and may either revolve backwards or forwards, according to the design of the screen.

It will be noticed that the tippler shown above does not stop automatically after a complete revolution has been made, but continues to revolve as long as pressure is kept upon the lever. There are now many patent tipplers at work which, when started, continue to revolve until a complete revolution has been made, and then stop automatically.

The corves run in at one end of the tippler and pass out at the other after being tipped. If a very large tonnage has to go over one tippler, it should be made of sufficient length to hold two corves, end to end.

Sorting and Cleaning the Coal.—Before the coal is sent to the purchasers it must be cleaned and sorted into various sizes and qualities. In some districts the large house-coal is picked out and loaded by hand, and the remainder separated into sizes by screens.

The smudge used for coke-making is usually washed to remove the dirt and other impurities, and it is now becoming the practice to wash the smaller sizes which are used for steam and gas coals.

Fixed-bar Screens.—These consist of flat spouts, having bottoms composed of steel bars; the bars are set in combs, and have spaces of any desired width between them. They dip from the pit bank to the waggons at the rate of 1 in 2 to 1 in 3. As the coal passes over the bars, the small slips through the spaces and the large passes over the bars. Thus, to make three sizes, slack, cobbles, and large, the upper portion of the screen would have bars fairly close together; in the second portion the bars would be spaced further apart, and the remainder of the screen would have a plate instead of bars. The large coal and cobbles would pass over the first portion of the screen, but the slack would fall through into a waggon; the cobbles would fall through the next set of bars into a second waggon, and the large coal would pass over the end of the screen into a third waggon.

Fixed-bar screens are now seldom erected at important collieries, as they do not separate the coal thoroughly, and offer no facilities for picking out the dirt.

Jigging Screens.—Fig. 190 shows a simple arrangement of jigging screen suitable for making three sizes of coal, namely, nuts, dust, and cobbles.

The screens themselves consist of iron pans, the screening being done over wire meshes or perforated sheets of iron. In the figure, *a* is the main screen, upon which the whole of the

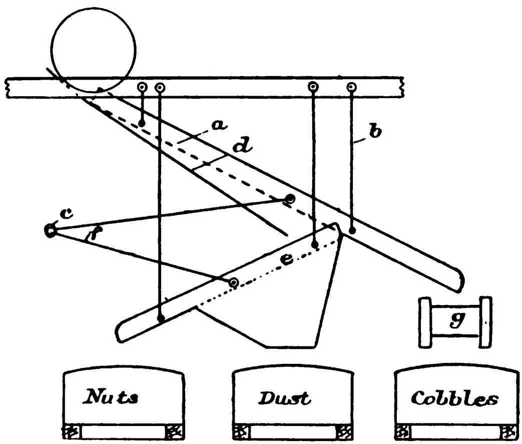


FIG. 190.—Jigging screens.

coal is tipped; it is hung from girders by the suspension rods *b*, and is given a reciprocating motion by means of the cam or eccentric *c*. The upper portion of this screen consists of a mesh sufficiently large to admit the nuts and dust; the lower portion is a steel plate, and acts only as a shoot. An iron shoot, *d*, is suspended below the mesh, and extends to the top of the lower screen, as shown in the figure.

The smaller screen *e* is also suspended on rods, and is vibrated by the eccentric *f*; the bottom of the upper portion of this screen is composed of a wire mesh, which allows the

slack to pass through, but not the nuts; the lower portion is of sheet iron.

The coal is tipped on to the top of the screen *a* and passes gradually down the screen, being well shaken about as it does so by the action of the eccentric. The nuts and dust pass through the mesh on to the shoot *d* and the cobbles pass over the end of the screen on to a picking band, *g*, and from thence to the waggon. The nuts and slack are conveyed by the shoot to the top of the smaller screen, where the slack passes through the mesh into the waggon and the nuts fall over the end of the screen on to another picking band, or direct into the waggon. There are many varieties of jigging screens; some vibrate sideways instead of endways, and the screens themselves can be arranged in many ways.

Picking Bands.—In appearance a picking band is similar to a long iron trough having a bottom composed of sheet-iron plates, which travels slowly along. The coal is tipped on to one end of the trough, and is conveyed slowly to the other end. Men and lads are stationed on either side of the belt, and pick out the dirt and other impurities from the coal as it is carried past them. The speed of the belts which form the bottom of the trough is usually from 30 to 50 feet per minute; and they are usually horizontal or very slightly inclined.

At many collieries in the Midlands belts are not only employed for picking out the refuse, but for picking out the various qualities of coal. For example, a seam may consist of several different qualities of coal—say house, steam, and gas. As the coal comes out of the pit the corves are tipped upon the belt, and a mixture of the coals covers the surface of the belt in a thin layer. The waggons are loaded on either side of the belt, and each is in charge of men and lads, who confine their attention to picking out one class of coal.

Thus one man might be filling a house-coal waggon, and would be stationed close to the belt, and to a waggon to be loaded with house-coal. As soon as a lump of house-coal is carried up to him, he lifts it off the belts and puts it into the

waggon. In this way all the large coal is picked off the belt, and the small which remains falls off the end to an elevator, by which it is raised and deposited on a jigging screen. Belts for this class of work are necessarily very long, some of the Derbyshire belts being as much as 300 feet in length by 4 feet 6 inches in width; they are set just above the tops of the waggons, which come close up to them on either side.

When belts are used for cleaning purposes only, their length is usually about 50 feet; the coal is first sorted into the various sizes by screens, and the dirt in each size picked out on an independent belt.

Picking belts are constructed of steel plates about $\frac{3}{16}$ inch in thickness, and 12 or 14 inches wide; these plates overlap

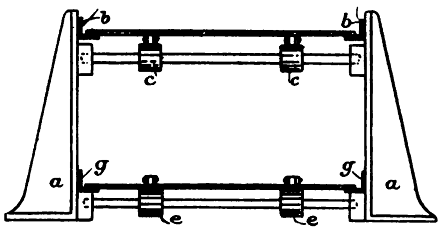


FIG. 191.—Section through a picking band.

each other, and are bolted or riveted to two or three sets of links.

The belts are driven by hexagonal drums, shaped to engage with the links. The method of carrying the belts is shown in Fig. 191, which is a section across one.

a, *a* are cast-iron brackets, fixed in pairs at intervals of 6 or 8 feet along the whole length of the belt. The steel angles *b*, *b* form the sides of the belt, and assist in carrying it and its load of coal. The rollers *c*, *c* are carried by the brackets, and support the belt under the links. The underside of the belt is carried by the angles *g*, *g* and rollers *e*, *e*.

Coal Washeries.—The approaching exhaustion of some of the best and cleanest seams in some districts is leading to

the introduction of costly and elaborate washing machines in order to prepare the dirtier coals which are being opened out for the market. For many years slack has been regularly washed preparatory to being coked, but it is only recently that the larger sizes have been washed for sale.

All washing machines depend for their action upon the difference in specific gravity between coal and the impurities from which it has to be freed. The specific gravity of coal is from 1·2 to 1·3, and of shale about 2·3. There are three classes of washing machines in general use, namely, *troughs*, *stirrers*, and *jiggers*.

Trough Washers.—If bodies of different specific gravities are allowed to sink in still water, they will not all reach the bottom at the same time, but the heaviest will arrive first, and the remainder follow in order of their specific gravities. Thus, if pieces of shale and coal were allowed to sink together, the shale, having the higher specific gravity, would reach the bottom first. If the water were not still, but were running at the rate of, say, 5 feet per second, and supposing the bind sank to the bottom in 2 seconds and the coal in 3, the bind would be carried by the current for a distance of 10 feet before it reached the bottom, but the coal would travel 15 feet, so that in this way the shale would be separated from the coal. It follows from this that the velocity of the current of water can be so arranged, by varying the inclination of the trough, as to carry the clean coal over the end, whilst the shale and other impurities fall to the bottom.

The trough washer in its original form consists of a wooden trough about 2 feet in width, 1 foot in depth, and 100 feet or so in length. It is divided at intervals by dams 3 or 4 inches in height, and has an inclination of about 1 in 25. Both water and slack are delivered into the top of the trough, and as the stream carries the slack down it is stirred up by men with shovels placed at intervals along the trough sides. The stream carries the washed slack over the end of the trough and deposits it in a hopper, whilst the shale falls to the bottom and is caught by the dams.

Two troughs are employed, being placed side by side, so that one can be cleaned out whilst the other is being used.

Washers of this description are rather costly in labour, and require a large quantity of water; there is also a danger of part of the very fine dust being lost.

The Murton Washer.—This machine is similar in principle to the ordinary trough washer, but is more elaborate, and, being automatic in action, requires much less labour.

The general design of the Murton washer will be understood from Fig. 192.

a is a travelling steel trough; it is about 60 feet in length, 18 inches in depth, and 3 feet in width at the top, which is

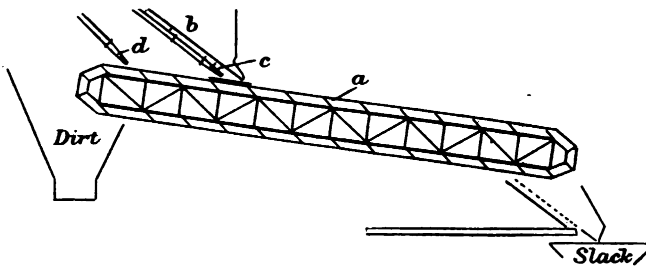


FIG. 192.—Murton washer.

rather wider than the bottom. This trough is made up of sections, each about 3 feet in length, and joined together by a watertight joint; at the end of each section there is a stopping 2 inches in height. The trough has an inclination, which can be varied to suit the coal which is to be washed, but averages about 1 in 20. It is carried by rollers, and travels uphill at the rate of 8 or 10 feet per minute; the driving arrangements are similar to those of an ordinary picking belt, except that the drums are very much larger. The slack to be washed is raised by an elevator to the hopper *b*, from which it is fed on to the belt.

A jet of water from the nozzle *c* meets the slack and carries it down the belt, whilst the dirt falls to the bottom, is caught by the stoppings, and carried uphill with the belt. A

second jet of water issues from the nozzle *d*, which is fixed a few feet above *c*; this stirs up the dirt on the trough and washes back any coal there may be among it. In this manner the washed coal is carried downhill by the water, and the dirt which settles on the bottom of the trough is carried up with it.

As the trough turns at its upper extremity the dirt falls off into a waggon, whilst the washed slack which is carried to the bottom of the trough by the water passes through a shoot, where the water is drained off into a hopper. The water passes through settling-tanks, where the sediment is deposited and the clear water is pumped back and used over again, with the addition of a little fresh, which must be added to make up the loss.

The slack is divided into various sizes before being washed, and each size is treated on a separate belt.

The Elliot Washer.—This is a form of trough washer in which the trough itself is stationary, but is provided with movable scrapers, which are fixed to chains and travel slowly uphill.

There are usually three troughs placed side by side; they are supplied with the slack which is to be washed by means of a revolving screen, which divides the slack into three sizes, and deposits each size in a separate trough. The washed slack is suspended in the water, and delivered by it at the lower end of the trough; the dirt falls to the bottom of the trough and is conveyed uphill by the scrapers, so that the clean coal is delivered at the lower end of the trough, and the dirt at the upper end.

Rotary Washers.—If a mixture of slack and shale is stirred together in water, the shale, being heavier, falls to the bottom before the coal; it is upon this principle that washers of the rotary class separate the coal from the dirt. Fig. 193 is a section through the Robinson washer, which is the best-known example of this class of machine. *a* is an iron pan, in the shape of an inverted cone. The vertical shaft *b* is fixed in the centre of the pan, and carries a crosshead, *c*, to which

are attached two sets of arms or stirrers, *d*. The shaft and arms are driven by an engine through bevel wheels, and make about ten revolutions per minute. *e* is a water jacket supplied with water at a slight pressure; it is an iron ring, surrounding the bottom of the cone, and provided with perforations in its inner circumference, through which the water issues to the interior of the cone. The slack to be washed is supplied

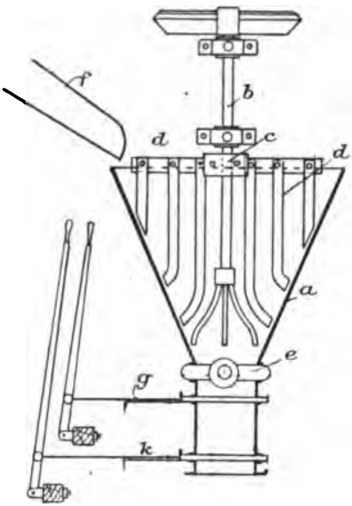


FIG. 193.—Robinson washer.

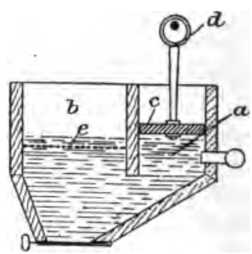


FIG. 194.—Jigger.

through the shoot *f* and falls into the cone, where it is agitated and thoroughly mixed with the water by means of the stirrers. The water entering through the perforations in the water jacket rises up the cone on account of its pressure, and flows over the top, carrying with it the clean coal, whilst the dirt gradually makes its way to the bottom. To discharge the dirt the lower slide *k* is closed, and the upper one *g* is opened; the dirt then falls into the space between the two slides. *g* is then closed and *k* opened, allowing the dirt to fall out into a waggon. By having the two slides it is

possible to remove the dirt without stopping the machine or losing the water.

The action, then, is briefly this: The water, in passing upwards, carries the clean coal with it and washes it over the top of the cone; whilst the dirt falls through the rising water, settles in the bottom, and is withdrawn by opening the slides. These washers are very simple in construction, compact, and cheap, and are much used for preparing slack for coking.

Jiggers.—Various designs of jiggers are employed in the elaborate and costly washing machines of the Luhrig and Humbolt type, a considerable number of which have lately been erected in the Midland and other coal-fields. These machines are designed to wash coal up to about 3 inches in diameter, as well as slack.

The coal is first carefully sized by revolving or other screens, and each size is dealt with by a separate jigger.

Jiggers were originally introduced for dressing metalliferous ores, and are still very largely used for that purpose. Fig. 194 shows the ordinary jigger; it consists of a rectangular box divided into two compartments, *a* and *b*, connected at the bottom as shown. *a* is fitted with a piston, *c*, which receives an up-and-down motion from the eccentric and rod *d*. The other compartment is divided horizontally by the sieve or strainer *e*. This sieve is slightly inclined, and is of just sufficiently small mesh to prevent the material which is to be washed from passing through it. The water is introduced through a pipe below the piston, and the coal to be washed is delivered on to the higher end of the sieve. Both the chambers *a* and *b* are kept full of water to a height of about 12 inches above the sieve. As the piston makes a down-stroke it forces water from *a* to *b*, thus raising the coal and shale lying upon the sieve, and allowing it to drop at the up-stroke. The piston makes from sixty to eighty strokes per minute, so that the material upon the sieve is rapidly lifted up and down, and as the coal is lighter than the shale, it is lifted higher and falls more slowly. The final result is, that coal and shale separate into

two layers, the coal of course being the uppermost. The coal is washed out at a wide opening about 12 inches above the sieve, and the shale is discharged at another opening situated a little lower down.

For washing the smaller sizes a felspar jigger is employed. In this case the openings in the sieve are sufficiently large to allow the material which is to be washed to pass through, but a bed of felspar about 3 inches in thickness is laid upon the sieve. The jigging action results in the shale finding its way below the felspar and through the meshes in the sieve, whilst the coal is washed over the opening in the side of the box as before.

The Baum Washer.—This machine differs from other washers in that the coal is washed before it is sized, whereas a complete sizing of the coal before washing is an essential feature of most other machines. By washing the coal before sizing only one jigger is employed instead of several. Another feature is that the water in the jiggers is pulsated by blasts of compressed air instead of by pistons. This does away with the eccentrics and pistons, and tends to make the machine simpler and less liable to derangements.

The Baum washer is comparatively new in England, but it has met with much success on the Continent, and is likely to be extensively used in this country in the future.

CHAPTER XXVII.

COKE-MAKING.

COAL is composed of fixed and volatile matters. By heating coal, the volatile matters are driven off, and the fixed remain in the shape of coke. Some coals, when heated, are resolved into a pasty mass which forms coke, quite unlike the coal in appearance, whilst others do not run together in this manner, but retain something like the original form of the coal. The former are termed caking coals, and can be used for coke-making, and the latter are known as non-caking coals, and will not coke. The coking properties of a coal cannot accurately be determined by its chemical analysis, but good coking coals contain from 3 to 4 per cent. of disposable hydrogen. By disposable hydrogen is meant the excess of hydrogen contained in a coal over that amount which can enter into combination with the oxygen there is present to form water.

The average composition of ordinary coking coal and of coke are as follows:—

	Coal per cent.	Coke per cent.
Carbon	60 to 85	85 to 90
Volatile matters ...	20 to 30	2½
Ash	3 to 5	5 to 10
Sulphur	0·5 to 1	0·5 to 2
Water	3 to 6	1 to 5

A certain amount of ash is necessary to give the coke mechanical strength. Sulphur is very harmful, especially in

the case of steel coke. Much moisture also should be avoided, as it is not only a source of loss commercially by displacing its weight of carbon, but a certain amount of the carbon has to be employed when the coke is used, to drive the moisture out in the shape of steam.

The chief varieties of coke are Furnace, Steel, and Foundry. Furnace and steel cokes should be very hard and porous, of steel-grey colour, clean and crystalline, and formed in columnar masses. They should be as free as possible from sulphur, and should not carry too much ash or water. Foundry coke should be very pure and compact.

Coke may be made either in ovens, into which air is admitted during the process of coking, such as the ordinary beehive oven, or in retort ovens, from which the air is entirely excluded.

Beehive Ovens.—The ordinary beehive oven is shown in section in Fig. 195. The oven itself is dome-shaped, usually 11 feet in diameter and from 7 to 8 feet in height. It is lined with a 9-inch thickness of firebrick, and built with ground fireclay instead of cement or lime. The floor is paved with hard red bricks.

The charging hole *a*, in figure, is a circular aperture about 12 inches in diameter, situated at the apex of the dome, and closed with a movable tile. The flue *b* is 9 inches in diameter, and connects the oven with the main flue *c*. The communication between oven and flue can be closed or regulated by means of the damper *e*. The doors *f* are 3 or 4 feet high, measured from the floor of the oven, and from 3 to $3\frac{1}{2}$ feet in width.

The oven floor stands about $2\frac{1}{2}$ feet above the level of the bench, and inclines slightly from back to front. The bench is a paved platform, about 20 feet in width, occupying the space between the front of the oven and the sidings.

Coke-burning.—The smudge to be coked requires washing, unless it is very pure and clean, or the coke will contain too much ash, and perhaps sulphur. In some places the smudge

or slack is ground and not washed, whilst at others it is both washed and ground. When both grinding and washing are done, opinions differ as to which should be done first. If the coal is ground first, it is difficult to wash, owing to its finely divided state; and, on the other hand, wet slack is more awkward to grind than when dry.

The ovens are charged through the holes in their tops, by mean s of iron hopper-bottomed waggons; the waggons are filled

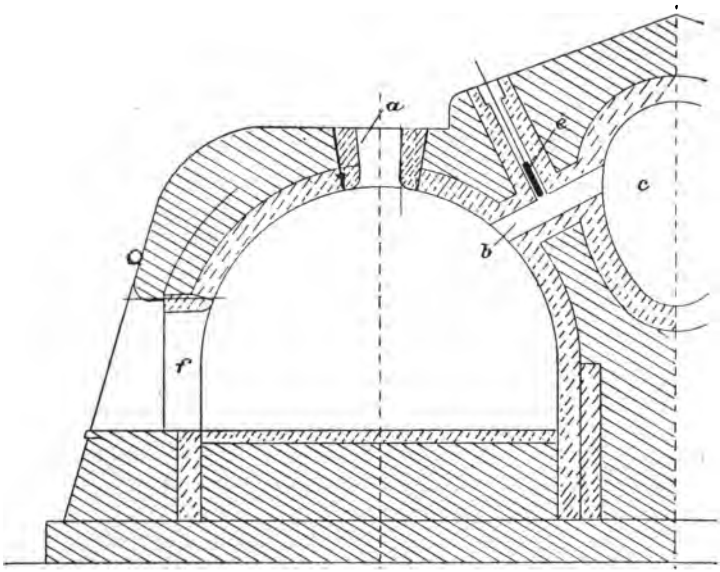


FIG. 195.—Beehive ovens.

from the main hopper, and run on rails laid on the ovens, to the oven which is to be charged. As soon as the waggon is exactly over the charging hole, a slide is withdrawn and the contents fall into the oven. The haulage on the ovens may be performed either by ropes or by a small locomotive.

After an oven is charged and the smudge levelled down, the door is built up with bricks and plastered over; a small hole is left at the top of the door to supply air for combustion.

This hole is enlarged or diminished as required, during the process of burning. The ovens are never allowed to cool, so that after one has been charged, it is quickly fired by the heat of the walls and of the adjacent ovens. The ovens are not drawn in blocks, but alternately, in order to keep the temperature as uniform as possible. The time required for coking varies somewhat with the nature of the coal, size of charge, and construction of the ovens. With 11-feet ovens of ordinary construction, charged with 6 or 7 tons of smudge, from 2 to 3 draws per week should be obtained.

When the coke is ready for drawing, it is cooled off with water before being drawn from the ovens; this is done by means of an iron pipe 12 or 14 feet long and about $\frac{3}{4}$ inch in diameter. This pipe is connected by a flexible hose to the water main which runs along the ovens, and water is sprayed on to the burning coke.

When ovens are drawn by hand, the contents are pulled through the door by means of long iron rakes and scrapers.

The heat and smoke given off during burning pass from the oven into the main flue, and are drawn under boilers and used to generate steam. The number of ovens per boiler varies very greatly at different collieries. At some collieries there are as many as 45 ovens to one boiler, whilst at others there are only from 10 to 15. Under average conditions, the heat given off from each coke oven will evaporate from 15 to 25 gallons of water per hour.

Beehive ovens are always built in a double row; two rows of ovens back to back, discharging their heat into one flue, is the usual arrangement.

Fig. 196 is an example of the manner in which beehive ovens can be arranged to heat Lancashire boilers.

Beehive ovens are sometimes built with a system of internal flues, either for heating the ovens themselves or the air used for combustion.

Mechanical Drawing.—The cost of labour on beehive ovens, when drawn by hand, is from 1*s.* 6*d.* to 2*s.* 6*d.* per ton of coke made. This cost may be reduced by drawing the

ovens by machinery in the following manner. A small engine and boiler propels itself upon a road laid alongside the ovens. The engine carries a shovel-shaped extractor at the end of a long arm. The arm is fitted with a rack, which engages with gearing on the engine, and can be run in and out or swung round at an angle. To draw an oven, the engine is stationed opposite the oven door, and forces the extractor under the coke, by the rack. As the extractor is withdrawn, it brings the coke out with it, owing to the barbed shape of the shovel. Sometimes the coke is drawn on to a belt which runs just below the oven doors, and delivers the coke on to a bar

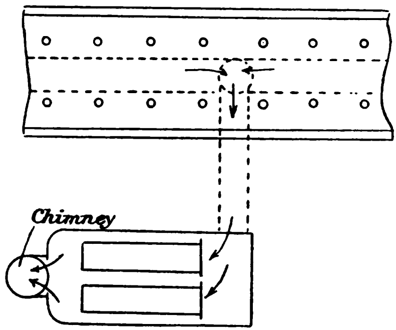


FIG. 196.—Beehive ovens with flues to boilers.

screen, from which it gravitates into waggons. These belts are not now used so much as formerly, owing to the cost of up-keep, caused by their great wear and rapid corrosion.

In beehive ovens the yield of coke is usually from 50 to 60 per cent. of the coal carbonized. Nearly the whole of the volatile matters and part of the carbon are consumed, and, except for the steam generated at the boilers, they are wasted.

Retort Ovens.—In making coke in retort ovens, the volatile matters are not burned away, but are driven off by heat transmitted to the coal through the walls of the ovens, and as all air is excluded from the coking chambers, there is no

combustion and no loss. Retort ovens are usually employed in conjunction with by-product recovery plants, by which the valuable products contained in the volatile matters are recovered.

There are now many batteries of retort ovens working in Great Britain, and in Germany they have entirely superseded beehive ovens.

A battery of retort ovens consists of a series of long narrow chambers, having flues between and under them; heat is transmitted from the flues through the walls of the ovens, which are constructed of firebrick, and must be very thin. The ovens are connected by an ascension pipe to an exhaust pump, which draws the gases off as they are generated by the decomposition of the coal. The gases are drawn through condensers and scrubbers, where they deposit tar and ammonia, and the incondensable gases which remain are led back to the ovens and burned at the flues to generate the heat necessary for coking the coal. There is practically no waste; the whole of the carbon should remain in the ovens in the form of coke, the heat which escapes from the ovens is taken under boilers to generate steam, the valuable by-products are recovered in the condensers, and any incondensable gas which remains after the ovens are heated is either burned under boilers, or used to drive gas-engines.

There are several different classes of ovens at work, all similar in principle, but differing greatly in detail. The flues between the ovens are sometimes horizontal and sometimes vertical; to the former class belong the Simon-Carvés and Semet-Solvay, and to the latter the Otto and Koppers ovens.

In the Yorkshire Coal-field the Simon-Carvés oven is more used than any of the others, and is giving very good results.

Fig. 197 is a cross-section of the Simon-Carvés oven. *a* is the oven itself, which is 32 feet 9 inches long, 8 feet 2 inches high, and from 20 inches wide at the front to 22 inches at the back. The oven is heated by the horizontal flues *b*, *b*; gas is brought from the by-product plant in pipes into these flues, and meets the air for combustion at various points, where it

ignites and heats the flues. The air for the combustion of the gas is heated before it enters the side-flues, by being drawn through arches in the oven foundations and along the sole-flue *f*. The heating is divided into two distinct zones, each independent of the other; the lower zone heats the lower part of the side walls and the upper zone the upper part. After the hot gases have heated the ovens they pass into a main flue, and from thence to a chimney, heating boilers on the way.

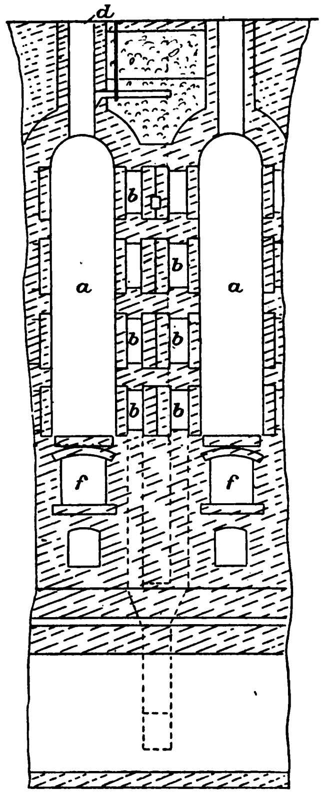


FIG. 197.—Simon-Carvés oven.

The ovens are charged either by means of a compressor, or through three charging holes, *d*, placed at equal distances along the oven tops; the charge is from 10 to $11\frac{1}{2}$ tons of coal. The gases are exhausted through an ascension pipe, having an outlet at the top of the oven.

The ovens are closed by doors at either end; these doors consist of firebrick built into an iron frame, and are raised vertically by chains and balance weights; they are luted with clay to keep air from entering the ovens. The process of coking is completed in about forty-eight hours. To empty the ovens, both doors are raised, and the coke is pushed out by a ram. The ovens are made a couple of inches wider at back than front, to enable the coke to be pushed out without jamming.

Compressors.—By these appliances the smudge is compressed and put into the oven in the form of a solid cake. They

are working successfully at several Yorkshire collieries, and result in a better coke.

Fig 198 is the front view of a compressor; that is, the view which is presented to the ovens. The whole apparatus travels on rails in front of the ovens, and is driven by an electric motor, taking its power from trolley wires, like an ordinary tramcar. *a* is the hopper, which is filled from the large storage hopper. *b* is the discharging ram; it is carried by a rack, and can be pushed forward by a motor. *c* is the chamber in which the cake is made; this chamber is a little less every way than an oven, and has one side which can be slackened a little to liberate the cake. The bottom of the cake-chamber is an iron plate, having a rack on its under side; spur-wheels driven by the motor engage with this rack, so that the bottom of the chamber can be racked forward, carrying the cake with it. *d* is a stamper; it is continually raised and dropped, travelling backwards and forwards along the whole length of the chamber at the same time.

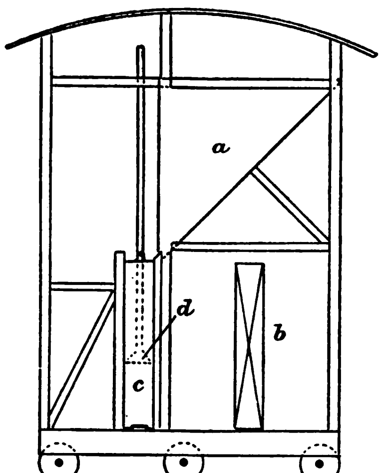


FIG. 198.—Coke compressor.

The process of charging and discharging an oven by means of a compressor is as follows:—

The compressor is run to the main hopper and takes a charge of slack into the hopper *a*; this hopper feeds the cake-chamber *c*, and as the cake-chamber is gradually filled, its contents are rammed by the stamper into a solid mass, water being added to make the smudge bind. The compressor is brought up to the oven which has to be discharged and is stopped with the ram *b* exactly in line with the oven. Both

the oven doors are then raised, and the ram is racked forward, pushing the coke out of the oven on to the bench behind it, where it is cooled with water and loaded up. After the discharging ram is racked back, the compressor is moved until the chamber *c* is exactly in line with the open oven. The door of the chamber is then raised, and the bottom plate is racked slowly forward into the oven, carrying the cake with it. When the whole of the cake is inside the oven, the doors are lowered, and the bottom plate drawn back, leaving the cake in the oven.

By-products.—Tar, ammonium nitrate, and benzol are the usual by-products which are collected from the volatile matters in the coal.

The gas given off when the coal is heated is drawn from the ovens by an exhaust pump. It first passes through the hydraulic main, where it is drawn through water and deposits part of its tar; it is next drawn through condensers, where it is further cooled and deposits more tar; it then goes through the exhausters, and is forced through scrubbers and washers, in which the ammonia is extracted; then through a second series, for the recovery of the benzol; and finally through an acid washer, which removes the last trace of ammonia.

The incondensable gas that remains is burned in the oven flues, and the surplus used to drive gas-engines or burned under boilers.

CHAPTER XXVIII.

ACCIDENTS.

THE following table shows the number of fatal accidents per thousand persons employed at the mines in Great Britain, classed under the Coal Mines Regulation Act, under different heads and extending over various periods :—

Period.	Underground workers.					Surface workers taken separately.	General : all persons employed above and below.
	Explosions of gas and coal dust.	Falls of ground.	Shaft accidents.	Miscellaneous.	All causes underground.		
10 years ending 1882	0·65	1·12	0·32	0·47	2·57	0·92	2·24
10 years ending 1892	0·32	1·00	0·19	0·50	2·01	0·96	1·81
Year 1893 ...	0·28	0·76	0·20	0·51	1·75	0·83	1·56
,, 1894 ...	0·54	0·77	0·16	0·32	1·79	0·79	1·59
,, 1895 ...	0·09	0·76	0·19	0·61	1·66	0·84	1·49
,, 1896 ...	0·30	0·76	0·14	0·42	1·62	0·87	1·47
,, 1897 ...	0·03	0·85	0·10	0·53	1·51	0·69	1·34
,, 1898 ...	0·05	0·77	0·11	0·44	1·37	0·88	1·27
,, 1899 ...	0·09	0·75	0·14	0·43	1·41	0·75	1·27
,, 1900 ...	0·07	0·79	0·13	0·45	1·44	0·70	1·29
,, 1901 ...	0·19	0·74	0·12	0·42	1·47	0·89	1·35
,, 1902 ...	0·09	0·68	0·15	0·44	1·37	0·69	1·23
10 years ending 1902	0·17	0·76	0·14	0·46	1·53	0·79	1·38

These figures show that the death-rate from accidents is gradually decreasing. During the ten years ending 1882, out

of every thousand workers underground over $2\frac{1}{2}$ were killed annually, whilst only $1\frac{1}{2}$ per thousand were killed during the ten years ending 1902.

Explosions.—Statistics show that there are now fewer colliery explosions than formerly, but that those that do occur are more extensive. Hence, the loss of life per explosion is now greater than it used to be. The decrease in the number of explosions is due to the better ventilation, the reduction in the use of blasting-powder, and the extended use of safety-lamps; whilst the greater death-rate per explosion is owing to the fact that mines are now larger, deeper, and drier than they were.

Until comparatively recently, it was thought that gas alone was responsible for explosions; but it is now proved beyond doubt that this is not so, and that coal-dust has played the most prominent part in many of the large explosions of recent years.

The fact that coal-dust alone could give rise to an explosion was doubted by many until recently, although it was universally acknowledged that if, in addition to the dust, a very small percentage of fire-damp were present, a disastrous explosion might be occasioned.

That coal-dust and air can cause all the effects of a violent explosion without the presence of gas is proved by the following:—

1. A violent explosion can readily be caused by the experimental firing of coal-dust and air.

2. Explosions of coal-dust and air have taken place on screens, etc., on the surface.

3. Violent explosions have taken place at collieries in which no gas has ever been seen, either before or after the explosion.

4. Many recent explosions have originated in the main intakes, in which dust in its most dangerous form was most likely to be found, and in which it was impossible for large volumes of gas to have been present.

Sources of Dust.—The downcast shaft of a colliery is, as a rule, closely surrounded by the screens. Consequently a large quantity of the fine dust that is always present on the screens is drawn into the mine with the air. This dust is carried along the roadways, and as the velocity of the current slackens, is deposited on the roof, floor, and sides of the intake airways. Much dust is also made whilst the coal is in transit from the workings to the shafts. The roads along which the coal is drawn are always more or less uneven, so that the lumps of coal are shaken and ground together, thereby forming dust. The corves, too, are frequently in bad repair, and allow small lumps of coal to fall out on to the road, where they become trampled and ground into dust. The heavier dust settles on the floor, and the lighter, which is the more dangerous, lodges on roof and sides.

As the main haulage roads are usually the intakes, it follows that the dust made in the transit of the coal settles in the intakes, in addition to the dust which is carried down the shaft from the screens.

A certain amount of dust is made in the workings by the breaking up and loading of the coal.

Dangerous Conditions of Dust.—The following are the factors which have the chief influence on the explosiveness of coal-dust and air :—

1. Nature of the dust : The dust from some seams is more dangerous than that from others.

2. Fineness: The fine dust is more dangerous than the coarse.

3. Dryness : The danger of coal-dust depends greatly upon its dryness. Air at a given temperature can only hold a given quantity of moisture in suspension ; but this quantity increases as the temperature rises. Air may enter a deep mine at a comparatively low temperature, but as it traverses the roadways its temperature increases, and so does its capacity for absorbing moisture. This being so, any moisture there may be present is taken up by the air, which dries everything it comes in contact with, including, of course, the coal-dust. For this reason

the coal-dust in deep mines is usually drier and more dangerous than in shallow ones.

4. Quantity: The quantity of dust in the air necessary to give rise to an explosion depends upon various circumstances; but the dust must be intimately mixed with the air in the shape of a cloud, though the cloud need not necessarily be very dense.

Action of a Coal-dust Explosion.—There is no actual difference, except in degree, between what is known as "combustion" and an "explosion." With the latter the combustion is extremely rapid, and if there is not ample room for the expansion caused by the heat and combustion, great pressure is generated. What actually happens when an explosion of coal-dust and air takes place is this: A cloud of fine dry coal-dust is raised in the air, either by a blown-out shot or by a small explosion of fire-damp. As soon as the dust is raised, it is instantaneously burned. Great heat is generated, causing the air to expand, and this expansion is increased by the addition of the gaseous matters formed by the burning of the solid particles of coal. As the space is confined, the result of this expansion is to cause great pressure, and to drive the flame along the road at a high velocity. As the flame advances, it raises fresh clouds of dust in front of it, upon which it feeds. Thus the flame rushes through the roads at ever-increasing pressure, raising a cloud of dust before it, and continuing until the shafts are reached or the supply of dust fails.

Coal-dust explosions usually originate in the main haulage roads, because it is there that the chief supplies of fine dry dust are found, and they are generally started by blasting operations. It will be noticed that the Explosives in Coal Mines Order, 1899, is drafted almost entirely with a view to guarding against the dangers of blasting in dry haulage roads.

Remedies.—To combat the dangers of coal-dust explosions, the dust may be removed at frequent intervals, or may be rendered innocuous by being kept thoroughly wet. There are, however, practical objections to both of these courses. It is

in many cases impossible to remove all the fine dry dust from the whole length of haulage roads, and to thoroughly wet the roads in some seams would set them "working" and add enormously to the cost of maintenance. Watering may be done by means of stand-pipes, hoses, or by water-barrels. At some collieries stand-pipes connected to mains containing water and compressed air are erected at intervals along the main roads, and send out a continuous and very fine spray. It has, however, been found that in order to wet the roads thoroughly these stand-pipes must be nearer together than is practicable. Water-barrels, as usually employed, are very inefficient, and are apt to interfere with the haulage. They are of no value unless they provide for the thorough wetting of the roof and sides, as it is there, and not on the floor, that the finest and most dangerous dust is found. In several instances explosions have been stopped by lengths of road happening to be naturally wet, and this has suggested the idea of keeping lengths of road thoroughly wetted. This expedient is not to be relied on, as there are cases on record in which explosions have passed over considerable lengths of wet road.

The best method of avoiding coal-dust explosions is to fire no shots in the main roads. If the provisions of the Explosives in Coal Mines Order are rigorously carried out, the risk of explosions from blasting are very small.

Whenever a large explosion takes place, it is found that the majority of deaths are caused by the effects of after-damp, and not by the force of the blast. Most explosions originate in the main intake, and travel in the direction of the shafts. This causes the main intakes to be filled with poisonous gases, and disarranges the ventilation by blowing out doors, stoppings, and overcasts. The men working at the coal face usually hurry into the main roads and are overcome by the after-damp; whereas if they remained in the workings until rescued, or came out by the returns, they might in many cases be saved.

The composition of after-damp has been found to vary very considerably, but the most important poisonous element is in all cases carbon monoxide.

The Pneumataphor.—To facilitate the exploration of mines after an explosion has occurred, apparatus have been devised by the use of which explorers carry the air they require with them, and are enabled to penetrate the most poisonous of atmospheres. One of the most recent inventions for this purpose is that known as the Pneumataphor. It consists of an indiarubber bag, about 24 inches by 20 inches by 3 inches, which is carried on the chest of the user, and is fitted with a mouthpiece. Inside this bag is a cylinder of perforated sheet iron, 8 inches in length by 3 inches in diameter, containing in a glass bottle a 25 per cent. solution of caustic soda; and below this is a cylinder 12 inches long containing oxygen at a pressure of 1500 lbs. per square inch.

To use the apparatus, the bag is strapped upon the chest of the user, who fixes the mouthpiece firmly into his mouth, and lightly clamps his nostrils with a small clip, to prevent air being drawn in through them. He then breaks the bottle of caustic soda by means of a screw provided for that purpose, and turns on a supply of oxygen. The whole of the breathing is done through the mouthpiece into the bag; the air which is breathed into the bag is polluted with carbonic acid gas which is absorbed by the caustic soda, and a small quantity of oxygen is added, making the mixture in the bag fit for respiration. Sufficient oxygen is carried for about 1½ hour's respiration, but if the wearer is exerting himself to any extent, this period is much shortened. Two smaller cylinders of oxygen may be carried instead of one, so that the explorer may know when half of his supply is exhausted.

In some of the apparatus of this class a helmet is worn instead of the mouthpiece and nose-clip. This prevents the mouthpiece being accidentally pulled out of the mouth, and the glass front protects the eyes of the wearer from smoke, if any should be present.

Although these appliances have been of great service in many cases, they are still very imperfect. The bags are very clumsy, and could hardly be used in rough low roads, and the carbonic acid gas is not completely eliminated from the

exhaled air, which gives rise to severe headaches on the part of the wearers.

Falls of Roof and Sides.—During the last ten years, falls of roof and sides have caused one-half of the deaths which have resulted from accidents underground, the mortality from this cause amounting to 0·76 per thousand workers underground. The accidents generally occur singly, and happen for the most part at the coal face. Thick seams, and seams lying at a high inclination, are generally more dangerous to work than thin and flat seams; the amount of danger also depends to a great extent upon the nature of the roof which overlies the coal. It does not by any means follow that the most dangerous work gives rise to the greatest number of accidents, a large proportion of the fatalities being caused by carelessness on the part of the workmen. The most efficient method of preventing accidents from falls is undoubtedly the introduction of systematic timbering. The timber at the face should be set at stated intervals, and not merely where it appears to be required at the moment; because the object of timbering is not only to support a bad roof, but also to prevent a good roof becoming bad.

One of the most dangerous operations in coal-mining is the withdrawal of props from goaves. This danger is greatly lessened by drawing the timber regularly, and before it has been left far behind, and almost disappears if the ringer and chain are properly employed, as then the timber drawer may stand under a good roof.

Shaft Accidents.—The fatalities caused by accidents in shafts during the last ten years amounted to 0·14 per annum per thousand persons employed, which is less than 1 fatal accident for 7000 employees. A portion of these accidents occur to men whilst being lowered or raised to or from their work, and the remainder to sinkers, or to men employed in repairing shafts, attending to pumps and similar work.

Considering the large number of men who are wound up

and down shafts every day, the number of accidents that happen is very small. The accidents that do occur are occasioned either by the engine-man overwinding, by the breakage of ropes or chains, or by men falling from the cages.

Detaching hooks (see Chapter XXIII.) save many lives, but they are of no avail if the cage is pulled into the head-gear at a high velocity, and in no case can they save the men in the descending cage, who, when an overwind occurs, are dashed violently into the pit bottom. There are now several different appliances in use by which engines are stopped automatically at the end of the wind, but arrangements of this kind are not at all general.

The breakage of winding-ropes or chains is rare, but occasionally serious accidents do arise from this cause, even when the tackle is of the best, and has been well cared for and properly examined. Wire ropes sometimes suffer from internal corrosion, and this may be very difficult to detect, the seat of the injury being in the inner wires. A very severe strain may be put upon the winding-rope if the engine-man suddenly checks the speed of the descending cage.

The weakest part of a rope is the capping; this should be renewed frequently, and a few yards cut off the rope at each renewal. The cage chains should be annealed every few months.

At many Continental and a few British collieries, the cages are fitted with appliances which wedge them to the conductors in the event of the rope breaking. These safety cages are generally employed in conjunction with timber conductors, though they can be applied with rope guides; they have not become popular in this country on account of their liability to come into action when the velocity of the cages is unusually great.

It is generally considered desirable to rely for safety upon the employment of ropes having a high factor of safety, and upon careful and frequent examination.

Miscellaneous Accidents.—About one-third of the

fatal accidents which occur underground are classified under this head. They may be subdivided as follows :—

1. Accidents on haulage road.
2. From suffocative gases.
3. From the use of explosives (see Chap. X.).
4. From eruptions of water.

Accidents on haulage roads are due to lads being run over whilst driving, and to persons being run over on engine planes and inclines. It is now becoming customary in large collieries to have a separate travelling road for the men, and to allow no one on the haulage roads except the men employed upon them. This is a very good practice, especially where the seam is steep. Where no separate travelling road exists, good manholes should be provided.

The fatalities occasioned by suffocative gases—leaving after-damp out of the question—are frequently due to the fumes given off from underground fires. These fires may arise either by accident, or by spontaneous ignition. Accidental fires may be caused by the use of furnaces or underground boilers, by the heating of brakes on hauling engines, by the careless handling of naked lights, or by the insufficient insulation of steam-pipes or electric machinery. No accumulations of oily waste should be allowed in engine-rooms, either on the surface or underground.

Spontaneous Ignition.—Some seams are extremely liable to spontaneous ignition, whilst in others it is never known to occur. As might be expected, gob-fires are almost entirely confined to the thick seams in which much small coal is left in the wastes. The principal agent in spontaneous ignition is the property that all coal possesses of absorbing oxygen from the air. The chemical action that this gives rise to is accompanied by the generation of heat, and the hotter the coal becomes, the greater is its affinity for oxygen, so that as the coal absorbs oxygen, it becomes heated, and as it becomes heated it absorbs still more oxygen and generates still more heat, until finally the temperature of ignition is reached, and the coal gives off dense smoke and bursts into flame. The

iron pyrites which some seams of coal contain was once thought to be the principal factor in spontaneous ignition, but it is probable that the chief part that pyrites play in the matter is to assist in the disintegration of the coal, thereby exposing fresh surfaces for the oxygen to act upon. Solid coal never ignites spontaneously, though fires may occur at the edges of pillars which are crushed by the weight.

Seams liable to gob-fires should be worked by the longwall retreating method, wherever it is possible to do so, as by this method of working the goaves are left behind. It is also most desirable to arrange the pit in such a manner that any district can be readily isolated by the building of two or three dams.

There are two methods of dealing with gob-fires. One is to seal them off by the erection of dams, and so extinguish them by cutting off the supply of oxygen; and the other is to cool them down with water and send the smouldering material out of the pit, filling in the space it occupied with sand or flue dust.

FIG. 199.—Dam against gob-fire.

Fig. 199 shows the class of dam that should be built to seal off a gob-fire. In building dams of this description, it is most important that the surrounding strata should be perfectly solid and free from breaks, as the dam is quite useless if air can get past it through the measures.

The dam shown in Fig. 199 is constructed of two brick walls, the one nearest the fire being convex in plan in order to resist the force of an explosion if one should occur in the vicinity of the fire. The space between the two walls is tightly packed with sifted sand and the outside of the outer wall kept well whitewashed.

A pipe should be taken through the whole of the dam and fitted on its outer end with a valve which allows the air or gas from within the dam to escape, but prevents the passage of air from the mine to the fire. The walls should be built in trenches cut right into the solid in roof, floor, and sides.

When fires occur in ordinary longwall workings it is usually impossible to dam them off, as there may be no solid ground in which to place the dams. In this case roads are scoured through the goaves to the seat of the fire, which is cooled down by water and filled out.

In a few instances carbonic acid gas has been generated, either by passing air over a coke and lime fire, or by the action of hydrochloric acid on limestone, and has been piped into the burning district which has been closed up. This procedure has not met with great success.

Eruptions of Water.—The Coal Mines Regulation Act provides that, "where a place is likely to contain a dangerous accumulation of water, the workings approaching that place shall not at any point within 40 yards of that place exceed 8 feet in width, and there shall be constantly kept at a sufficient distance, not being less than 5 yards in advance, at least one borehole near the centre of the working and sufficient flank boreholes on either side."

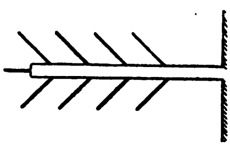


FIG. 200.—Boring in advance to prove old workings.

The arrangements that might be adopted when a longwall face approached old workings filled with water are indicated in Fig. 200. The heading and boreholes must be driven continuously as the face advances, and in no case must the boreholes be less than 5 yards in advance of the heading or the heading less than 40 yards in advance of the face. If the pressure of the water in the old workings was at all considerable, boreholes of a much greater length than 5 yards should be provided.

The long holes are usually drilled by means of auger drills and rotary machines worked by hand. The lengthening rods should be composed of pipes, and the drills hollow, to enable the hole to be cleaned by pumping water through the rods. When the water pressure is expected to be great, boring should be conducted through a pipe firmly wedged into the hole and provided with a valve, so that when the water is tapped it can be shut off.

In every case when boring against water, long wooden plugs should be in readiness. These plugs should be fitted with crossbars, to enable the men to force them into the hole against the pressure.

In some cases ancient submerged workings have been found to have become almost entirely closed up, and to contain but little water.

Dams.—It is sometimes necessary to shut off a feeder of water by closing the roadways along which it flows by means of dams.

These dams must be very substantial, as they may have to resist enormous pressure, and their failure might be disastrous to both life and property. The total pressure upon a dam depends upon the area exposed to the water, and upon the head of the water. For example, what will be the total pressure upon a dam 9 feet 6 inches wide by 8 feet high, the height to which the water will rise when the dam is built being 230 yards?

$$\text{Area of dam, } 9\tfrac{1}{2} \times 8 \times 144 = 10944 \text{ sq. inches}$$

$$\text{Pressure per square inch, } 230 \times 3 \times {\cdot}434 = 299{\cdot}5 \text{ lbs.}$$

$$\text{Total pressure in tons, } \frac{299{\cdot}5 \times 10944}{2240} = 1463{\cdot}3 \text{ tons}$$

The best dams to resist very great pressure are those constructed of solid timber. Each piece of timber must be carefully shaped and of taper form, the thicker end being exposed to the pressure. The timber should be built in the form of an arch, and the joints well wedged up from behind. Dams of this construction, though very efficient, are extremely costly, and are only built in exceptional cases.

Fig. 201 shows a masonry dam. It consists of two rows of brick arching, built of hard bricks, set in cement. Each ring should be built independently of the others, and a small space left between, which should be filled in with cement as the dam is being built. The site of the dam must be very carefully chosen, and a trench excavated for it beyond all breaks. It is very difficult to make a satisfactory joint between the top of a

masonry dam and the roof, as a slight settlement takes place, especially if the dam is built up quickly and the joints in the brickwork are not kept very thin. An iron pipe should be built into the bottom of the dam, to allow the water to flow through whilst the dam is being built, and to enable the workmen engaged in the erection to pass out when it is completed. A small pipe should also be built into the dam near the top, to allow the air to escape as the space behind the dam becomes filled with water, after which it can be gradually closed to allow the dam to take the full pressure by degrees.

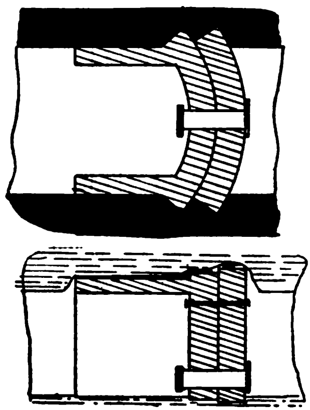


FIG. 201.—Masonry dam.

Diseases to which Miners are liable.—Coal-mining is not considered an unhealthy occupation, in fact the rate of mortality among coal-miners compares favourably with that of most classes of manual labour.

Phthisis.—Lung troubles are usually prevalent among men who work in an atmosphere impregnated with dust, but coal-miners, as a class, suffer little from consumption and similar maladies, because coal-dust is free from hard grit. Men who work in metalliferous and ganister mines are very liable to lung disease, especially where rock drills are employed, as the dust made by the drills is composed of sharp, hard particles, which cut the lungs.

Ankylostomiasis.—Extensive outbreaks of this disease have recently occurred in Westphalia; it is extremely contagious, and drastic methods have had to be resorted to in order to cope with it. This disease takes the form of intestinal worms, the growth of which are favoured by the prevalence of a warm, moist atmosphere. When an outbreak occurs, the affected

persons should be carefully isolated, and all the underground workers periodically subjected to medical examination. All collieries should be provided with portable sanitary appliances, placed in convenient places underground, and their use insisted upon.

Nystagmus.—This is a disease which affects the nerves of the eyes; it is more prevalent among those who work by the light of safety-lamps than among those who use naked lights. The cause of nystagmus is thought to be either the constant dazzling of the eyes by the bright concentrated light of safety-lamps, or the strain put upon the muscles of the eyes by work such as holing. The symptoms of this disease are a twitching of the eyes and impaired sight.

CHAPTER XXIX.

ELECTRICITY.

ELECTRICITY is in itself so large and important a subject that a detailed treatment cannot be attempted in this volume, but the employment of electricity in mining is now so general that a few notes on the subject seem desirable.

The exact nature of electricity is not known, but it is evidently a condition of matter which exists to a greater or less extent among the atoms and molecules of which all bodies consist. Electricity in its quiescent state can do no work; it is only when its equilibrium is disturbed that power can be obtained from it. In like manner, water in itself has no power, but if raised and allowed to fall to its original level, power can be generated by leading it over a water-wheel or through a turbine. Electricity must not be regarded as a source of power, but only as a means of transmitting power; and the power given off by the motor is always less than that put into the dynamo.

Electric Terms.—The following are the most important of the terms used in electrical engineering.

The volt is the unit of pressure, potential, or electromotive force (E.M.F.).

The coulomb is the unit of quantity.

The ampère is the unit of current.

The ohm is the unit of resistance.

The watt is the unit of power.

The Board of Trade Unit (B.T. Unit) is the unity of energy.

The Volt.—The pressure at which electricity is generated is measured in volts, in the same manner in which pressure of steam or water is measured in pounds per square inch. The usual pressure generated at collieries is from 500 to 600 volts.

The Ampère.—The rate of flow of electricity is measured in ampères. Water is said to flow at so many gallons per minute; electricity at so many *coulombs* per second. One coulomb per second equals one ampère, hence if 5000 coulombs per minute flowed through a circuit, the current would be $\frac{5000}{60} = 83{\cdot}3$ ampères. For commercial purposes the quantity flowing per hour is taken as the unit, and quantities measured in *ampère hours*.

The Ohm.—The resistance offered to the passage of electricity is measured in ohms. When a given quantity of water passes through a pipe it encounters a certain amount of resistance; the amount of resistance depends chiefly upon the quantity of water and size of pipe. In like manner when electricity passes through a wire it encounters resistance, which is measured in ohms. The resistance depends upon the size and length of the conductor, the material of which it is composed, and upon the quantity of electricity passing, but does not vary with the voltage or pressure of the current.

The Watt.—Mechanical power is measured in foot-pounds per minute and in horse-power; electrical power is measured in watts. A current of one ampère at a pressure of one volt gives out power equal to one watt; thus ampères multiplied by volts equal watts. 746 watts equal one electrical horse-power. So that a current of 40 ampères at a pressure of 500 volts equals $\frac{500 \times 40}{746} = 26{\cdot}8$ H.P. One kilowatt is 1000 watts.

Board of Trade Unit.—An electric supply of 1000 watt-hours is known as a Board of Trade Unit (written B.T. Unit). This is really a commercial term, the electric light and power stations supplying their customers at so much per B.T. Unit.

Example.—If a current of 60 ampères at a pressure of 400 volts were employed for 24 hours, what is the total

number of units consumed, and what is the cost per horse-power per hour at $2\frac{1}{2}d.$ per B.T. Unit?

$$\text{Watts, } 60 \times 400 = 24000$$
$$\text{Watt hours, } 24000 \times 24 = 576000$$
$$\text{Units consumed, } \tfrac{596000}{1000} = 576$$
$$\text{Horse-power, } \frac{60 \times 400}{746} = 32{\cdot}17$$
$$\text{Units per hour, } \tfrac{576}{24} = 24$$
$$\text{Cost per hour, } 24 \times 2{\cdot}5d. = 60d.$$
$$\text{Cost per E.H.P. per hour, } \frac{60}{32{\cdot}17} = 1{\cdot}86d.$$

One B.T. Unit = $\frac{1000}{746}$ = 1·34 H.P. working for one hour.

Electrical Conductors.—Some materials offer very little resistance to the passage of electricity, whilst others offer so much as to altogether prevent it from flowing under ordinary pressures. The former materials are termed conductors, the latter are known as non-conductors or insulators. All the metals are conductors, though in varying degrees, while india-rubber, silk, oil, porcelain, mica, ebonite, glass, etc., are insulators. Water, wood, and several other substances are classed as semi-conductors, as they allow electricity to pass through them, though not very readily.

The Dynamo.—Electricity, when employed on a large scale, is generated by means of dynamos. The dynamo or electric generator depends for its action upon the fact that electric currents are generated whenever the lines of force, which pass from one pole of a magnet to the other, are cut by a conductor.

A magnet is anything which possesses the power of attracting iron or steel or other magnetizable bodies. There are two kinds of magnets, *Permanent* and *Electro.* The former are made of hard steel, and retain their magnetism for a considerable period, and the latter owe their magnetism to an electric current, and are magnets only so long as the electric current passes round them.

If a length of insulated wire is coiled round a bar of soft

iron, as shown in Fig. 202, this bar becomes an electro magnet whenever a current of electricity is sent through the wire. One end of the bar is known as the "North pole," and the other as the "South pole." "Lines of force" flow from the ends of the magnet, passing from the North to the South pole, as shown in figure. These "lines of force" are really currents of magnetism. If the magnet is bent into the shape of a horse-shoe, the lines of force will flow from one pole to the other, and the space between the ends of the magnet through which these lines of force pass is known as the magnetic field.

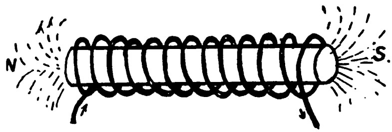


FIG. 202.—Electro magnet.

Whenever a wire is moved through a magnetic field in such a manner as to cut the lines of force, an electric current is induced, which flows through the wire.

In Fig. 203, N and S are the two poles of a magnet; consequently, lines of force pass through the magnetic field

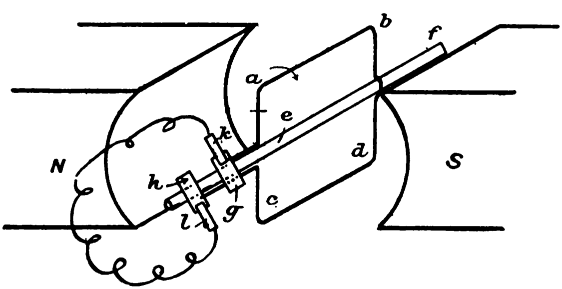


FIG. 203.—Principle of the dynamo.

between them. *abcd* is a wire frame, mounted on the spindle *ef*, by which it can be rotated. One end of this frame terminates in an insulated copper ring, *g*, and the other terminates in a similar ring, *h*. Brushes of copper, *k* and *l*, are kept in contact with the rings, and from these brushes the

electric current is led off as it is generated. When the frame is in the position shown in Fig. 203, no lines of force are cut; but if it makes half a turn in the direction of the arrow, the part *ab* descends through and cuts the lines of force, whilst the part of the frame *cd* ascends through them. This cutting of the lines of force generates an electric current in the frame, which passes from one brush through the external circuit, where it does work, and back to the machine through the other brush.

If the frame is rotated through the other half of the revolution, *cd* descends through the lines of force and *ab* ascends. This induces a current through the frame, but in the opposite direction, so that for each complete revolution of the frame two currents of electricity are generated, flowing through the frame in opposite directions. Machines of this class are known as alternators.

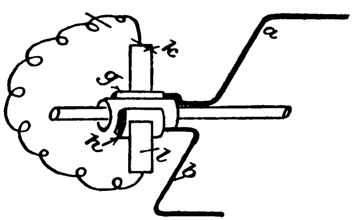


FIG. 204.

Alternating currents are changed to direct by means of an apparatus known as a "commutator." The principle upon which this is constructed is shown in Fig. 204. The ends of the frame or coil are not joined to copper tubes as in Fig. 203, but each end is connected to one half of a split tube, each half being insulated from the other and from the spindle. In Fig. 204, the end of the coil *a* terminates in one insulated half-tube, *g*, and the end of *b* in the other half, *h*. The brushes *k* and *l* are set exactly opposite each other, so one presses on the one segment and the other on the other segment.

During one half of a revolution the current through the frame travels in one direction, and is reversed during the other half, but the current in the opposite direction is taken by the opposite brushes, because the segments have turned round, so that the reverse current, being taken by the opposite brushes,

results in a continuous current. Or, in other words, as soon as the direction of the current changes, the connection to the external circuit is also changed, and the one change negatives the other, so the current travels through the outer circuit in the same direction as before.

For the sake of simplicity, the frame or armature shown in Fig. 204 has only one coil of wire, but in practice armatures are constructed of many coils, each insulated from the others, and each connected to a separate insulated bar on the commutator.

The main organs of a dynamo are shown in Fig. 205. *a*

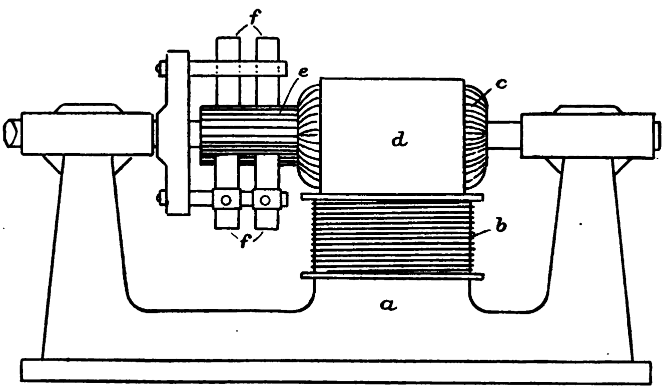


FIG. 205.—Dynamo.

is the magnetic yoke connecting the magnetic coils *b*; *c* is the armature revolving in the magnetic field between the field magnets *d*; *e* is the commutator, which changes the currents from alternating to direct; *f*, *f* are the brushes through which the current is led round the coils and external circuit.

The armature is rotated at a high speed by means of an engine and belt, the pulley on the engine being large and that on the dynamo small, in order to increase the speed of the latter.

In the apparatus shown in Fig. 203, the armature *abcd* is rotated between the poles of permanent magnets, but in large

dynamos electro magnets are employed, as shown in Fig. 205.

As has already been explained, electro magnets are formed by sending a current of electricity through coils wrapped round an iron core. The coils and core are shown at *bd*, Fig. 205, and the electric current which passes through them may be supplied either by a small dynamo, in which case the machine is said to be separately excited, or, as is more usual, the current which passes through the coils may be taken from the machine itself. When the armature is revolved, the slight amount of residual magnetism in the pole pieces causes an electric current to be generated in the coils of the armature, which is collected by the brushes and allowed to pass through the coils encircling the soft iron magnets, and the magnetic field becomes stronger and stronger, until the field pieces are saturated.

There are three different methods of winding dynamos—Series, Shunt, and Compound.

Series Machines.—The connections of a series dynamo are shown in Fig. 206. The current passes from the commutator through one brush, round the external circuit where the work is done, and back round the coils to the other brush. The whole of the current passes round the coils, so that in series machines the voltage given out varies with the current, because, when more current is generated, more passes round the field magnets, which are thereby strengthened.

Shunt Machines.—The coils of these machines are magnetized by being wound with a great length of fine wire, and only part of the current goes round them, the remainder serving the external circuit. It follows, therefore, that if the resistance on the external circuit is increased, more current flows through the shunt, thereby strengthening the field and increasing the voltage.

Compound Machines.—These machines are wound both in series and by shunts. In series machines, when the speed is constant, the voltage rises as the external current increases; in shunt machines the voltage falls as more current is employed,

so that by combining the two windings on one machine a constant voltage can be maintained under varying loads.

Fig. 207 illustrates the winding of a compound machine. The thick line represents the series coil, and the thin one the shunt. As the current leaves the + brush it splits, and a small portion of it goes through the shunt, and the remainder through the series coils; both series and shunt coils are taken round the field magnets, but the series coils only form the external circuit. If the load increases and more current is required, the voltage due to the series coils is increased, but that due to the shunt coils falls, and *vice versâ*. In this manner compound machines give a constant voltage under varying loads, if they are run at a constant speed.

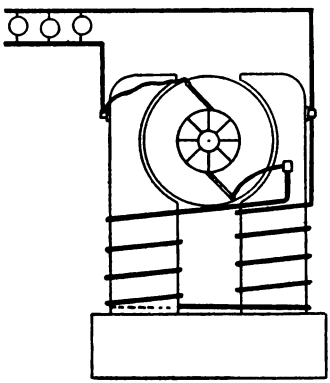


FIG. 206.—Series-wound machine.

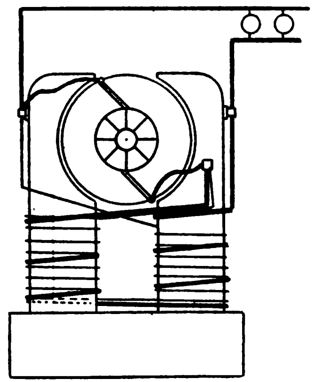


FIG. 207.—Compound-wound machine.

Compound-wound dynamos are the best for colliery work, as, by keeping their speed constant, a constant voltage is generated under the varying loads which occur when coal-cutting machines, hauling engines, etc., have to be driven.

Driving Dynamos.—Three methods of driving electric generators are in general use.

(*a*) By low-speed engines, through belts or ropes.

(*b*) By high-speed engines, coupled direct.

(*c*) By steam turbines.

The two latter methods are now generally preferred, though low-speed engines and belt drives have considerable advantages as regards simplicity and smooth running.

Steam turbines are now coming more to the front; they are extremely compact, and require little attention. They run at a very high speed—from 3000 to 30,000 revolutions per minute. When worked in conjunction with condensers they are not uneconomical as regards steam consumption.

Cables.—The electric current is conveyed from the dynamo to lamps, or motors, by insulated cables; for continuous currents two cables are required—one to take the current from the dynamo, and the other to conduct it back in order to complete the circuit. If the insulation on the conductors were destroyed, and the bare cables allowed to touch, what is known as a "short circuit" would be formed, and the whole or part of the current would go back to the generator through the point of contact. Short circuits may also occur in the dynamos or motors, and may give rise to heat and cause the insulation to burn.

The resistance that different metals offer to the passage of an electric current varies very greatly; silver has the least resistance, and is, therefore, the best conductor. Copper is almost universally employed for electric conductors, as it offers low resistance to the current, and is very durable. Taking the resistance of silver as 1, the resistance of copper is 1·01, and of iron 6·25; so that, under similar conditions as to length, a conductor of iron must have more than six times the area of one of copper to offer the same resistance to a current of electricity. Although the price of copper is much more than six times the price of iron, copper cables are the more economical, because, being so much lighter, they are cheaper to fix, and, being so much smaller in diameter, a less weight of insulation is required for a cover of equal thickness.

Cables are insulated with rubber, ozokerited tape, or with some material impregnated with oil; when used in damp places they should be protected by a lead covering, and, if exposed to rough usage, armoured by being wrapped round with galvanized wire.

The two cables may be either separate or concentric; a section through a cable of the latter type is shown in Fig. 208. *d* is an internal copper strand forming one cable; *b* is a layer of insulation to prevent leakage of current from one cable to the other; *c*, *c* are copper strips forming the second cable; *a* a second layer of insulation; and *e* wire armour.

On the surface cables are carried by being hung from poles, to which they are attached by insulators; down shafts they may be carried side by side in a grooved wooden casing, as shown in Fig. 209. The cables fit tightly into the grooves, a cover is nailed over them, and the whole spiked securely to the shaft side.

In the workings cables are usually hung from insulators

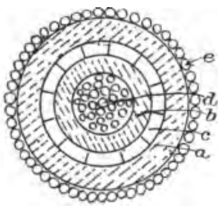


FIG. 208.—Concentric cable.

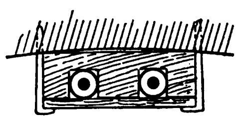


FIG. 209.—Methods of taking cables down shafts.

nailed on to the timbers; they should be hung loosely and allowed to form loops, so that a fall of roof would drag them down, and not break them. Sometimes both cables are hung on one side of the road, but it is better to have one on either side.

Size of Cables.—The loss of pressure in the cables depends upon their length, and upon the number of ampères carried per square inch sectional area of copper. The usual allowance of area is at the rate of one square inch of copper for 1000 ampères; this leads to a loss of about 2½ volts per hundred yards. If the cables are too small, the loss in voltage, and consequently power, may be serious; the power being expended in heating the cables.

Large cables are made up of a number of small wires twisted into strands; their size is expressed in the number

and gauge of the wires of which they are composed; thus a $\frac{7}{16}$ cable means a cable composed of seven wires each of No. 16 standard wire gauge (S.W.G.)

Cables are usually made up of wire of from Nos. 12 to 22 standard wire gauge.

The following table gives the sectional area of wires of various gauges, together with their carrying capacity at the rate of 1000 ampères per square inch :—

Number of standard wire gauge.	Sectional area in square inches.	Capacity at 1000 ampères per square inch.
12	0·0085	8·5
13	0·0066	6·6
14	0·0050	5·0
15	0·0041	4·1
16	0·0032	3·2
18	0·0018	1·8
20	0·0010	1·0
22	0·0006	0·6

Cables are usually constructed of 3, 7, 19, or 37 wires.

From this table the size of cable necessary to carry a given current can be determined.

Example.—What size cable would be required to carry a current of thirty-five ampères?

Area of copper in square inches, $\frac{35}{1000}$ = 0·035 square inch.

If the cable has seven wires, the area of each will be—

$$\frac{0·035}{7} = 0·005$$

The gauge of a wire 0·005 square inches in area is No. 14, so that a $\frac{7}{14}$ cable is required.

Motors.—The construction of a motor is similar to that of a dynamo, and machines built for dynamos will run as motors, or *vice versâ.*

A dynamo generates current when its armature is revolved,

whereas a motor is supplied with current, which causes its armature to rotate. The rotation is produced by the action of the lines of force in the magnetic field upon the coils in the armature.

The magnetic currents tend to turn the coils until they coincide with the lines of force, but as soon as one coil is pulled into this position, another takes its place, and is pulled round in its turn, and so on, the result being a continuous rotation of the armature.

Motors, like dynamos, may be wound in series, shunt, or compound.

In series motors the speed varies with the load, hence they are only suitable for constant loads. The "torque" or turning power of a series motor is greatest at the moment they are started, so that they are suited for starting against a heavy load, which is maintained at a fairly constant pressure after it is set in motion, such as pumping or hauling up an incline.

The starting torque of a shunt motor is low, but they run at a fairly uniform velocity against loads varying within certain limits. Compound-wound motors start against a heavy load by reason of their series coils, and maintain a uniform velocity by virtue of their shunt coils.

Distribution.—Where a large amount of power is generated at a central station, and used to supply numerous lamps and motors, as is usually the case, the whole current should not be generated by a single engine and dynamo, but by several. For example, if a maximum load of 800 horse-power had to be provided, five machines might be employed, each driven by a separate engine, and each capable of supplying 200 horse-power. This would allow one machine for reserve in case of repairs or break-downs. If only one large machine were erected, there would be no provision for accidents, and the large machine would have to run at times when only a small amount of power was required, consequently with less efficiency.

The cables from the various machines are led to the main switch-board, from whence the current is distributed among the different circuits.

Switches.—The current is turned on or off the different circuits by means of switches. One design of double-pole switch is shown in Fig. 210. One cable is attached to the terminal *c*, where it is broken off and attached to *a*. The space between *a* and *c* is bridged by the bar *e*, as shown in the figure. If the current is to be switched off, the handle is pushed sharply in the direction of the arrow; this moves the bar clear of the terminal *c*, and the contact is broken. As this is a double-pole switch, both wires must be disconnected; the manner in which this is done is shown in Fig. 210. The switch is mounted on a slate base, the cables being connected from behind.

Cut-outs.—If through any accident the current should rise

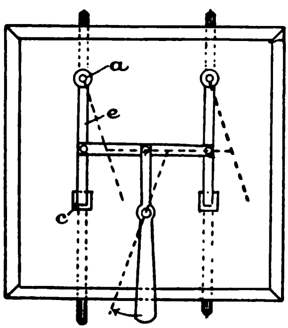


FIG. 210.—Double-pole switch.

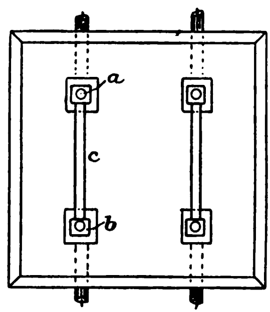


FIG. 211.—Fusible cut-out.

much beyond the normal, the insulation on the machines may be burned by the heat generated. To prevent this, cut-outs are placed in each circuit. A cut-out of ordinary form is shown in Fig. 211. The cable is broken off at *a* and reconnected at *b*, the space between *a* and *b* being bridged by a tin or lead fuse-wire *c*. This wire is of sufficient size to carry the maximum current that should pass; but, if this maximum is exceeded, the heat generated by the passage of the current through the wire melts it and cuts out the circuit.

Magnetic cut-outs are also employed, the contact being broken by the action of an electro-magnet when the current becomes excessive.

Starting Switches.— In order to start a motor gradually and without shock, resistances are employed. These resistances may consist of thin wire coiled into spirals. When the motor is started, the current is passed through all the coils, and the resistance they offer to its passage lowers the electro-motive force, as part of the power is spent in heating the wires; so that the whole power of the current is not turned into the motor. The resistance coils are gradually thrown out of the circuit by moving a lever, so that the potential gradually increases.

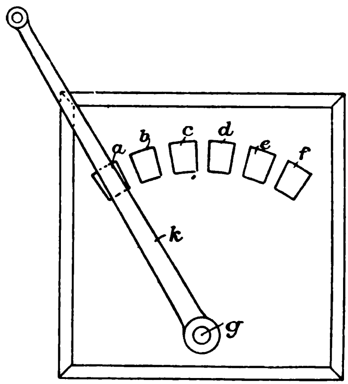


FIG. 212.—Starting switch.

In Fig. 212, *a*, *b*, *c*, etc., are contact blocks connected to the coils. In the position shown in figure, the current enters at *g*, passes through the bar *k*, contact block *a*, and through the whole of the resistances to the motor. By moving the switch to *b*, some of the resistance coils are thrown out, more are thrown out by moving it to *c*, and so on, until all the coils are out of circuit, and the motor gets the full pressure when the bar is moved to *f*.

Electric Lamps.—There are two classes of electric lamps, incandescent and arc. The former are usually employed for lighting engine-houses, offices, pit bottoms, etc., and the latter for large open spaces, such as sidings.

Incandescent lamps consist of a thin thread of carbon enclosed in a glass bulb from which the air has been extracted. When the electricity flows through this thread it raises it to an intense heat on account of the resistance offered to the passage of the current; this causes the lamp to glow. The carbon is not consumed owing to the absence of oxygen in the bulb. Incandescent lamps are commonly made of small candle-

power, 16 candle-power being a common size, but they are also constructed for very high powers.

1 horse-power will supply current for ten 16 candle-power lamps. Their usual voltage is from 100 to 200.

Arc lamps consist of carbon rods about $\frac{1}{2}$ inch in diameter. The rods are brought together and then separated by a space of about $\frac{1}{4}$ inch. A continuous discharge is then maintained through the space, causing a bright light. As the arc burns the carbon rods are consumed, and means are provided for feeding the rods forward as their length is shortened. 1 horse-power provides current for about 1000 candle-power when arc lamps are employed.

Systems of Wiring.—There are several methods by which electric energy is distributed. The most common method is by the two-wire system, arranged either in parallel or series.

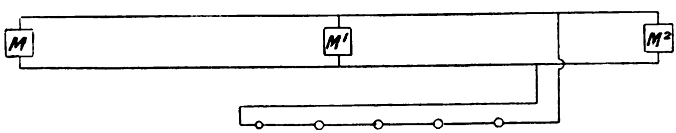


FIG. 213.—Wiring in parallel and series.

In Fig. 213 the two motors M[1] and M[2] are connected in parallel, and the five lamps in series with each other. If the dynamo is generating electricity at a pressure of 500 volts, the motors being connected in parallel, each receive a current of 500 volts; but the lamps, being in series, divide the pressure among them, so that the pressure on each lamp is 100 volts only. Motors are always arranged in parallel, but if it is desired to run 100 volt lamps on a 500-volt circuit, they may be arranged in series as shown. If one lamp fails, the others which are connected in series with it go out also.

Polpyhase Plants.—Three-phase alternators have been recently put down at several collieries, and their application is likely to extend. Their chief advantage is that they have no commutators, and can give off no sparks.

Three cables are employed instead of two, as with continuous currents, but they are smaller in sectional area. The machines themselves are very simple in construction; the motors can start against a heavy load, and do not race and burn up if they are suddenly relieved of their load.

Dangers of the Employment of Electricity in Mines.—The chief danger introduced by electricity in mines is the possibility of sparks from brushes or switches lighting any gas there may be present. There is also a possibility of a short circuit causing a fire. If the plant is well arranged, this danger is not great; but in several cases fires have been caused in this manner. Another source of danger is the risk of shock to those handling the apparatus. Under ordinary conditions a shock from a current of 500 volts would do little harm, but under exceptional circumstances men have been killed by shock from a current at a considerably lower voltage.

Size of Electric Machinery.—Electric machinery should always be made of ample size for the work it has to do, otherwise there is constant trouble from over-heating, which results in a final breakdown.

The size of motor for a given duty may be calculated as follows:—

Find size of motor to drive a pump delivering 240 gallons per minute against a head of 500 feet.

$$\text{Horse-power in water, } \frac{240 \times 10 \times 500}{33000} = 36{\cdot}36$$

Taking the efficiency from motor to water raised at 75 per cent., the horse-power of motor must be $\frac{36{\cdot}36 \times 100}{75}$ $= 48{\cdot}48$

$$\text{Watts required, } 48{\cdot}48 \times 746 = 36166$$

Assuming E.M.F. at motor to be 450 volts, amps. required, $\frac{36166}{450}$ $= 80{\cdot}4$

Size of motor required, 450 volts and 80·4 amps.

Cables.—Sectional area of cable in square inches at 1000 amps. per square inch, $\frac{80 \cdot 4}{1000} = 0 \cdot 0804$.

Area of each wire if nineteen are employed, $\frac{0 \cdot 0804}{19} = 0 \cdot 0042$

The standard wire nearest to this in size is No. 15, so that the cables should be $\frac{19}{15}$.

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